#903221382

ID 88068219

TN
413
.W2
S65
1994

TABLE OF CONTENTS

1.0 INTRODUCTION 1-1

1.1 SUMMARY OF PROPOSED PROJECT 1-1
1.2 BACKGROUND 1-3
1.3 PREVIOUS WORK AND ONGOING ACTIVITIES 1-4
1.4 EXISTING PERMITS 1-6
1.5 EXISTING FACILITIES 1-7
1.6 PROJECT PROPONENT 1-7
1.7 COMPANY REPRESENTATIVES 1-8

2.0 PROJECT OVERVIEW 2-1

2.1 PROJECT LOCATION AND ACCESS 2-1
2.2 PROJECT AREA AND OWNERSHIP 2-1
2.3 PROJECT SCHEDULE 2-2

3.0 DETAILED PROJECT DESCRIPTION 3-1

3.1 UNDERGROUND MINING PLAN AND METHODS 3-1

3.1.1 Determination of Ultimate Mining Areas 3-2
3.1.2 Primary Development 3-2
3.1.3 Development Ground Support 3-3
3.1.4 Mining Method Determination 3-3
3.1.5 Transverse Stoping with Cemented and Uncemented Rockfill 3-4
3.1.6 Longitudinal Stoping with Delayed Waste Fill 3-6
3.1.7 Drift and Waste Fill 3-6
3.1.8 Stope Stability and Support 3-7
3.1.9 Stope Drilling and Blasting Procedures 3-10
3.1.10 Ore and Waste Transfer Procedure 3-11
3.1.11 Backfill Production and Placement 3-12
3.1.12 Subsidence Control and Monitoring 3-15
3.1.13 Equipment Requirements/Consumables 3-18
3.1.14 Mine Ventilation 3-19
3.1.15 Explosives Storage and Handling 3-20

100
2001000
200
2.5

Section Page

3.2 BACKFILL QUARRY OPERATION 3-20

3.2.1 Quarry Area and Ownership 3-20
3.2.2 Summary of Project Activities 3-20
3.2.3 Project Schedule 3-21
3.2.4 Geology 3-21
3.2.5 Site Topography 3-22
3.2.6 Mining Operations 3-22
3.2.7 Reclamation 3-26

3.3 FIRE-FIGHTING EQUIPMENT/EMERGENCY RESPONSE 3-27
3.4 POWER/WATER REQUIREMENTS 3-27
3.5 TECHNICAL STUDIES 3-28

3.5.1 Rock Mechanics 3-28
3.5.2 Geochemistry of Underground Mine and Backfill Quarry 3-29

3.6 WASTE ROCK DISPOSAL AREAS 3-29
3.7 ORE TRANSPORTATION 3-30
3.8 COUNTY ROAD RELOCATION 3-33
3.9 NOXIOUS WEED CONTROL 3-34
3.10 LAMEFOOT EXPLORATION 3-35
3.11 SURFACE DISTURBANCE 3-36
3.12 RECLAMATION 3-37

3.12.1 Introduction 3-37
3.12.2 Reclamation Schedule 3-38
3.12.3 General Reclamation Procedures 3-40
3.12.4 Reclamation of Drill Sites/Roads 3-47
3.12.5 Drill Hole Abandonment 3-50
3.12.6 Reclamation of Disturbances Associated With Underground Exploration 3-50
3.12.7 Temporary Closure Plan 3-53
3.12.8 Post-Closure Monitoring 3-55
3.12.9 Public Safety 3-56
3.12.10 Noxious Weed Control Plan 3-56
3.12.11 Underground Reclamation 3-57
3.12.12 Backfill Quarry Reclamation 3-59

Section Page

3.13 WATER MANAGEMENT AND MONITORING 3-62

3.13.1 Grouting 3-63
3.13.2 Surface Water Controls 3-64
3.13.3 Underground Water Use 3-66
3.13.4 Alternate Blasting Compounds 3-67
3.13.5 Water Treatment 3-67

3.14 HYDROLOGIC MONITORING PLAN SUMMARY 3-78
3.15 OPERATIONAL SAMPLING OF BACKFILL AND WASTE ROCK 3-80

3.15.1 Backfill Quarry 3-80
3.15.2 Underground Waste Rock 3-80

3.16 RECORDKEEPING AND REPORTING 3-80

4.0 REFERENCES 4-1

LIST OF TABLES

TABLE 1 COUNTY PERMITS REQUIRED FOR THE KETTLE RIVER OPERATIONS
TABLE 2 STATE PERMITS REQUIRED FOR THE KETTLE RIVER OPERATIONS
TABLE 3 FEDERAL PERMITS REQUIRED FOR THE KETTLE RIVER OPERATIONS
TABLE 4 MAXIMUM VOID HEIGHT FOR PLANNED UPPER LAMEFOOT STOPES AFTER BACKFILLING
TABLE 5 CROWN PILLAR WIDTH AND THICKNESSES FOR UPPER STOPES
TABLE 6 MAXIMUM POSSIBLE SURFACE SUBSIDENCE ABOVE NEAR-SURFACE STOPES (INCHES)
TABLE 7 MAXIMUM POSSIBLE SURFACE SUBSIDENCE-INDUCED STRAINS ABOVE NEAR-SURFACE STOPES
TABLE 8 LAMEFOOT PROJECT COMPONENT EQUIPMENT LIST
TABLE 9 LAMEFOOT OPERATIONS FUEL, OIL, LUBRICANTS, EXPLOSIVES AND CHEMICALS - 1993 CONSUMPTION DATA
TABLE 10 PROJECT-RELATED WATER PRODUCTION AND USE
TABLE 11 ESTIMATED AREA OF SURFACE DISTURBANCE FOR KETTLE RIVER PROJECT COMPONENTS
TABLE 12 COVERSOIL SUMMARY
TABLE 13 FERRY COUNTY NOXIOUS WEED LIST

LIST OF FIGURES

FIGURE 1 GENERAL LOCATION MAP
FIGURE 2 LAMEFOOT PROJECT SITE ACCESS AND REGIONAL TOPOGRAPHY
FIGURE 3 SITE TOPOGRAPHY SHOWING EXISTING DRILL ROADS, FACILITIES, AND EXPLORATION/DEVELOPMENT WORKINGS
FIGURE 4 OBLIQUE VIEW OF UNDERGROUND PHASE I EXPLORATION/DEVELOPMENT PROGRAM
FIGURE 5 OBLIQUE VIEW OF UNDERGROUND PHASE II EXPLORATION/DEVELOPMENT PROGRAM
FIGURE 5A OBLIQUE VIEW OF UNDERGROUND PHASE II PROGRAM ON BLM LAND
FIGURE 6 FACILITY LAYOUT
FIGURE 7 LAND OWNERSHIP AND CLAIMBLOCK BOUNDARIES
FIGURE 8 MINE DECLINE DEVELOPMENT TECHNIQUES
FIGURE 9 OBLIQUE VIEW OF UNDERGROUND EXPLORATION AND COMMERCIAL DEVELOPMENT PROGRAM
FIGURE 10 MINING SEQUENCE LAMEFOOT PROJECT, LONG SECTION LOOKING EAST (ZONES 1 THROUGH 4)
FIGURE 11 LAMEFOOT PROJECT, LONG SECTION LOOKING EAST (ALL ZONES)
FIGURE 12 PLAN VIEW OF TYPICAL LAYOUT OF HOLES DRILLED IN SUBLEVEL DRIFTS
FIGURE 13 STANDARD LAMEFOOT BOLTING PATTERN WASTE HEADINGS
FIGURE 14 LAMEFOOT PROJECT TRANSVERSE STOPING, PRIMARY STOPING
FIGURE 15 LAMEFOOT PROJECT TRANSVERSE STOPING, SECONDARY STOPING
FIGURE 16 LAMEFOOT PROJECT LONGITUDINAL STOPING
FIGURE 17 LAMEFOOT PROJECT DRIFT AND WASTEFILL
FIGURE 18 STOPING AND BACKFILLING METHODS, LAMEFOOT PROJECT, LONG SECTION LOOKING EAST (ZONES 1 THROUGH 4)
FIGURE 19 CONCEPTUAL DIAGRAM OF SUBLEVEL STOPING WITH LONG-HOLE AND RING DRILLING AND BLASTING METHODS
FIGURE 20 LAMEFOOT PROJECT CEMENTED ROCKFILL PLANT CONCEPTUAL LAYOUT

FIGURE 21 GROUT MIX PLANT
FIGURE 22 SURFACE MONITORING STATION MAP
FIGURE 23 SURFACE MONITORING STATION
FIGURE 24 BACKFILL QUARRY AND COUNTY ROAD DIVERSION
FIGURE 24a RELOCATED COUNTY ROAD SECTION 1 PLAN AND PROFILE
FIGURE 24b RELOCATED COUNTY ROAD SECTIONS 2 AND 3 PLAN AND PROFILE
FIGURE 24c RELOCATED COUNTY ROAD SECTIONS 4 AND 5 PLAN AND PROFILE
FIGURE 24d RELOCATED COUNTY ROAD SECTION 6 PLAN AND PROFILE
FIGURE 25 BACKFILL QUARRY SECTION 26100N
FIGURE 26 BACKFILL QUARRY SECTION 26900N
FIGURE 27 PROPOSED FINAL RECLAIMED TOPOGRAPHY OF THE LAMEFOOT FACILITIES AREA
FIGURE 28 LOCATION MAP FOR TYPICAL ROAD SECTIONS
FIGURE 28a TYPICAL ROAD SECTIONS, CONTOURS DURING OPERATIONS
FIGURE 28b TYPICAL ROAD SECTIONS, FINAL RECLAIMED CONTOURS
FIGURE 29 CONCEPTUAL DIAGRAM OF PORTAL #1 ADIT CLOSURE
FIGURE 30 CONCEPTUAL DIAGRAM OF PERMANENT SHAFT CLOSURE
FIGURE 31 CONCEPTUAL DIAGRAM OF OLD TIMER'S ADIT A CLOSURE
FIGURE 32 CONCEPTUAL DIAGRAM OF OLD TIMER'S ADIT B CLOSURE
FIGURE 33 CONCEPTUAL DIAGRAM OF BACKFILL CROSS CUT CLOSURE
FIGURE 34 ROADSIDE INTERCEPTION DITCH PLAN VIEW
FIGURE 35 ROADSIDE INTERCEPTOR DITCH - REACH 1
FIGURE 36 ROADSIDE INTERCEPTOR DITCH - REACH 2
FIGURE 37 ROADSIDE INTERCEPTOR DITCH - REACH 3
FIGURE 38 ROAD CULVERT, ROADSIDE INTERCEPTOR DITCH REACH 3
FIGURE 39 OPTIONAL CHANNEL EROSION CONTROL MEASURES, ROADSIDE INTERCEPTOR DITCH REACHES 1 AND 2
FIGURE 40 QUARRY RETENTION BERM - WOLFE CAMP ROAD DRAINAGE
FIGURE 41 ION EXCHANGE SYSTEM BLOCK FLOW DIAGRAM

FIGURE 42 BIOLOGICAL DENITRIFICATION SYSTEM WITH SRBC TECHNOLOGY, BLOCK FLOW DIAGRAM
FIGURE 43 BIOLOGICAL DENITRIFICATION SYSTEM WITH HIGH RATE FIXED FILM FILTER TECHNOLOGY, INTERMEDIATE BACKWASH BLOCK FLOW DIAGRAM
FIGURE 44 ELECTROCHEMICAL ION EXCHANGE SYSTEM, BLOCK FLOW DIAGRAM
FIGURE 45 PRETREATMENT SYSTEM, LAMEFOOT PROJECT
FIGURE 46 LAMEFOOT MINE WATER QUALITY MONITORING BY YEAR
FIGURE 47 LAMEFOOT MINE PROJECT AND MONITORING EVALUATION SCHEDULE

LIST OF APPENDIXES

APPENDIX A KETTLE RIVER MILLING FACILITY
    APPENDIX A-1 DESCRIPTION OF MILLING OPERATIONS
    APPENDIX A-2 EXISTING ENVIRONMENT - MILLING SITE
APPENDIX B OTHER ELEMENTS OF KETTLE RIVER OPERATIONS
    APPENDIX B-1 KETTLE MINE
    APPENDIX B-2 OVERLOOK MINE
    APPENDIX B-3 KEY MINES
    APPENDIX B-4 K-2 PROSPECT
    APPENDIX B-5 OTHER ECHO BAY ACTIVITIES
APPENDIX C BLM RECORDATION NUMBERS OF UNPATENTED CLAIMS AT THE LAMEFOOT PROJECT
APPENDIX D DOE MONITORING REQUIREMENTS FOR KETTLE RIVER OPERATIONS
    APPENDIX D-1 SPECIAL CONDITIONS OF DOE MONITORING REQUIREMENTS
APPENDIX E GEOTECHNICAL STABILITY REPORT
APPENDIX F RESULTS OF MARCH 1993 NOISE SURVEY
APPENDIX G LAMEFOOT PROJECT WATER QUALITY ANALYTICAL DATA
    APPENDIX G-1 ALLUVIAL MONITOR WELLS LF-1 AND LF-2
    APPENDIX G-2 INFILTRATION POND LF-I
    APPENDIX G-3 DATABASE TABLES AND GRAPHS FOR LF-I, LF-1, AND LF-2
APPENDIX H LAMEFOOT GEOCHEMISTRY REPORT, ECHO BAY MINERALS

APPENDIX I BULKHEAD DESIGN CALCULATIONS
APPENDIX J HYDROLOGIC MONITORING PLAN
  APPENDIX J-1 SOP NO. 1 - EQUIPMENT DECONTAMINATION
  APPENDIX J-2 SOP NO. 2 - GROUNDWATER MONITORING WELL AND PIEZOMETER INSTALLATION
  APPENDIX J-3 SOP NO. 3 - FIELD PACKER TESTING
  APPENDIX J-4 SOP NO. 4 - AQUIFER PUMPING TESTS
  APPENDIX J-5 SOP NO. 5 - WATER LEVEL MEASUREMENT
  APPENDIX J-6 SOP NO. 6 - MONITORING WELL SAMPLING
  APPENDIX J-7 SOP NO. 7 - SAMPLE HANDLING, DOCUMENTATION AND ANALYSIS

1.0
INTRODUCTION

This document serves to modify the April 1993 Modification of the Lamefoot Plan of Operations (POO) with supplemental information from July 1993 (Echo Bay 1993) submitted to the Department of Interior, Bureau of Land Management (BLM), Spokane District. This modification refines the description of underground mining activities associated with commercial production of gold ores from federal land under the jurisdiction of the BLM and associated activities on adjacent private land. Also described as an integral part of the project is the transportation of the ore to the existing Kettle River milling facility (Key Mill) and the excavation of backfill to be placed underground. The processing of ores mined at the Lamefoot deposit is described in Appendix A.

1.1 SUMMARY OF PROPOSED PROJECT

The Lamefoot Project is one component of the Kettle River Operations, a gold mining and processing complex located in Ferry County, Washington (Figure 1). The Lamefoot Project has been an ongoing exploration program since 1989, evaluating a mineralized gold deposit for commercial production. The first phase of exploration utilized surface drilling to discover and delineate this deposit. The second phase, currently in progress, employs underground exploration mining and drilling techniques. This modification to the existing POO (Echo Bay 1993) describes the commercial mining phase scheduled to begin in July, 1994, and potential additional surface exploration to explore for the continuation of gold mineralization.

The commercial mining phase of the Lamefoot Project involves the extraction of ore from the mineralized body using underground mining techniques and transportation of the ore via state and county roads to the existing, previously permitted milling facility. Approximately seventy percent of the underground mining will take place on private land and thirty percent will occur on BLM land. Eleven acres of the mining area is on private land and 4 acres are on public land. Possible future expansions may affect up to an additional 5 acres. As of March 1994, the areal extent of the underground workings, including the access tunnel, totals 7.3 acres.

The mining itself will be accomplished by a long hole, open-stoping mining technique. This method entails drilling blast holes between existing mining levels within the ore body and blasting the rock so that it may be accessed from lower levels. The ore will then be loaded into underground trucks and will be transported to the surface. Highway trucks will haul the ore by state and county roads to the processing facility (Figure 2).

The ore will be processed by the conventional carbon-in-leach method at the existing Key Mill. Processed ore will be permanently stored in a zero discharge-lined tailings facility after extraction of gold. The tailings impoundment is located on private land and is scheduled and permitted for enlargement from 1993 through 1995. Process waters are treated in a cyanide destruction circuit before recycling for use in the milling circuit. Milling, metal extraction, and tailings disposal are discussed in more detail in Appendix A.

Waste rock from underground development will either be deposited directly as backfill in the completed workings or deposited by underground trucks in a surface disposal area at the Lamefoot site. Waste rock will be placed in such a way as to prevent acid rock drainage. Most of the waste rock generated during both the commercial mining phase and the current exploration phase will be hauled underground upon completion of mining in individual areas. This practice will provide ground support and mitigate any potential for acid formation. The waste rock remaining on the surface after mining will be used to backfill and recontour the portal, and will be revegetated with an approved seed mixture after placement of stockpiled topsoil. The existing buildings will be dismantled and the office/shop site and other surface disturbance will be revegetated. Drill roads and pads from previous and future surface exploration programs will be reclaimed.

Throughout the mine life, studies will be conducted that will characterize the potential for acid generation in the underground workings and the potential for subsidence. Appropriate mitigation measures have been developed and will be refined to address the findings of these monitoring programs.

Mined-out areas within the Lamefoot underground workings will be backfilled with rock, as required to minimize subsidence and infiltration of surface water into the mine. Rock will be obtained from the temporary waste rock stockpile and from an excavation on private land south of the portal. The county's Wolfe Camp Road will be relocated to an alignment through this excavation at the end of mine life.

No additional surface disturbance on public land during the commercial mining phase at Lamefoot is anticipated after completion of the ongoing and permitted exploration phase. However, if additional surface exploration is required, minor surface disturbance may be associated with drilling or operations activities, affecting 7 to 12 acres.

1.2 BACKGROUND

The Lamefoot Project is an element of the Kettle River Operations which has been a producer of gold from several mining locations within Ferry County, Washington. Kettle River is operated by Echo Bay Minerals of Edmonton, Alberta, and Denver, Colorado.

An exploration program at the Lamefoot site has been ongoing since 1989 on both private and federal land. A Notice of Operations for exploration was initially submitted to the BLM in September of 1989 for surface drilling on federal land. Several modifications were made to that notice and, in 1992, a POO (Echo Bay 1992a) was submitted in anticipation that surface disturbance would exceed 5 acres on property managed by the BLM.

As part of the scope of the POO (Echo Bay 1992a), an underground exploration project was designed to confirm the results of surface drilling, establish feasibility of commercial production of gold ore, and obtain bulk samples of the ore for processing studies. That underground program was considered as a cumulative effect to the surface drilling program and was addressed in an Environmental Assessment (EA; BLM 1992a) supporting the POO (Echo Bay 1992a). All of the surface disturbance associated with the underground exploration program has been confined to private land. The underground exploration program is currently in progress and is scheduled to continue through July 1994.

The POO was modified in April 1993 to describe underground mining and related activities associated with commercial production of gold ores.

In 1988, a Final Environmental Impact Statement (FEIS; DOE 1988) was issued for the Kettle River Project which, at that time, included two underground mines (Overlook and Kettle River; Appendix B) and a mill/tailings storage facility. Currently, both of these mines are idle and ore is supplied to the mill by the two small open-pit mines, Key East and Key West. A FEIS (USFS 1992) was completed in 1992 for the Key Project Expansion.

The proposed Lamefoot Mine will be an integral element in the Kettle River Operation upon depletion of reserves from the Key East and Key West mines. The Lamefoot ore is necessary for the continued operation of the facility. This ore will be processed in the previously permitted Key Mill and will be stored in the tailings facility which is scheduled for previously permitted enlargement in two phases during 1993 - 1995.

1.3 PREVIOUS WORK AND ONGOING ACTIVITIES

From 1989 to 1992, extensive surface drilling was conducted at the Lamefoot Project by reverse-circulation rotary and coring techniques. Exploration drilling occurred throughout the land holdings but focussed on the primary area of mineralization in a zone about 1,500 feet long and up to 140 feet wide. A total of 280 holes were drilled on the surface, 59 of these are located on federal land.

Roads were built to accommodate access by drilling equipment and support vehicles. All of the surface disturbance to date on federal land is associated with this drilling program. Figure 3 shows the location of existing drill roads on public and private land. Areas of disturbance are summarized in Section 3.10. A detailed description of these roads and the drilling can be found in the POO (Echo Bay 1992a) and EA (BLM 1992a).

Currently the Lamefoot deposit is being explored, assessed, and developed by an underground exploration/development program consisting of two phases. Phase I includes the following elements:

- Development of surface facilities to support the underground activities

- Mining of a tunnel to access the ore deposit
- Mining of four levels along the mineralized body
- Mining of a ramp to interconnect the different levels
- Mining of a ventilation raise to the surface
- Mining of crosscuts and raises along the ore to establish lateral and vertical continuity of the mineralization
- Providing working space and facilities necessary for normal mining operations
- Processing of some or all of the mined ore for metallurgical study

Figure 4 shows an oblique view of the completed Phase I underground program.

Phase II was implemented to further explore, delineate and develop specific areas of the mineralized body in preparation for commercial mining. The primary elements of Phase II include:

- Mining of two additional levels in between initial drifts on ore
- Development of access drifts

Figure 5 shows an oblique view of the completed exploration and development after Phase II. Figure 5A shows the same oblique view delineating only the underground workings located on BLM land.

As of May 1994, the total material extracted during the Phase I and II underground exploration program is approximately 260,000 tons of waste rock and 114,000 tons of ore.

Additional exploration will be conducted for extensions of the known mineralization and for new deposits in the immediate area. This exploration will utilize both drilling and drifting underground and may require drilling from the surface. The possible surface drilling program is described in Section 3.10. Underground exploration will be conducted by extending existing workings toward prospective areas, utilizing techniques described in Chapter 3.0. Underground core drilling will test these target areas with electric/air-powered drilling rigs. The precise areas of continued underground exploration cannot be determined until the Phase II underground exploration program is complete. This program is likely to continue through the commercial mining phase until mine closure.

1.4 EXISTING PERMITS

The EA (BLM 1992a) and March 1992 version of the POO (Echo Bay 1992a) detail and document the scope of previous and ongoing surface and underground exploration activities of the project. A Record of Decision approving these activities was issued by the BLM on July 28, 1992.

Previous environmental documentation and regulatory approvals of activities for the Kettle River Operations include a FEIS for the Key Mill and the Overlook and Kettle mines. This document was completed and approved in 1988 under the authority of the Washington Department of Ecology (DOE 1988).

A FEIS (USFS 1992) completed in 1992 addressed environmental impacts of the Key Project Expansion and supported a Plan of Operations (Echo Bay 1992b) for two small, open-pit mines. Participating agencies for the Key Expansion FEIS were the U.S. Forest Service, the Washington Department of Ecology, the BLM, and the Washington Department of Natural Resources.

Various state and county permits have been acquired over the life of the Kettle River Operations. These permits, along with federal approval, are summarized in Tables 1 through 3. DOE Permit No. 8033 is attached as Appendix D.

The permits currently held for the Lamefoot Project include waste discharge, septic, building, and burn permits. Permits to be secured include an air quality permit and a water appropriation permit.

1.5 EXISTING FACILITIES

Existing facilities at the Lamefoot Project include the ongoing underground workings associated with the exploration/development program, as described in Section 1.3.

Site surface infrastructure consists of an office, shop, dry (change house), core storage facility, supply storage area, and parking. Services include an electrical substation, compressor, two water wells, and a septic field. Also present at the site are waste rock disposal areas, an infiltration pond, and topsoil stockpile areas. Diversion ditches have been constructed at the site to control storm water runoff. The site layout showing these previously approved and constructed facilities is presented on Figure 4 in the March 1992 version of the POO (Echo Bay 1992a).

The actual site layout on private land is very similar to the plans described in the above documents. However, certain refinements in the locations of the buildings, portal, well and other elements were incorporated into the final design due to site conditions encountered during construction. Figure 6 shows the current facility layout. No changes in the planned surface disturbance have occurred on BLM land.

Roads associated with surface drilling activities occur in the immediate vicinity of the site on both fee and BLM land. The location and characteristics of these roads are fully described in the March 1992 version of the POO (Echo Bay 1992a).

1.6 PROJECT PROPONENT

The operator of the Lamefoot Project is Echo Bay Minerals Inc., a subsidiary of Echo Bay Mines Ltd. of Edmonton, Alberta. Principal offices in the U.S. are located in Denver, Colorado, and Reno, Nevada. The Lamefoot Project is operated out of the Kettle River Operations office, located at the mill site near Republic, Washington.

1.7 COMPANY REPRESENTATIVES

Company representatives for the Kettle River Operations are:

Mr. Jerry McCrank, General Manager
Echo Bay Minerals Inc.
921 Fish Hatchery Road
Republic, Washington 99166

and

Mr. Walt Hunt, Superintendent, Technical Services
Echo Bay Minerals Inc.
921 Fish Hatchery Road
Republic Washington 99166

2.0
PROJECT OVERVIEW

2.1 PROJECT LOCATION AND ACCESS

The Lamefoot Project Area is located along the western flank of the Kettle River mountain range of northeastern Washington State in north central Ferry County. The area is approximately 120 miles northwest of Spokane, Washington, seven miles northeast of Republic, Washington, and twenty miles south of Curlew, Washington. The project site is accessed from State Highway 21 which runs in a north/south orientation between Republic and Curlew. Access through the Lamefoot area from State Highway 21 is via Ferry County's Wolfe Camp Road.

Figure 3 shows the general topography of the Lamefoot area. Figure 2 identifies site location, access, and shows the boundary of the study area considered in the baseline studies supporting this POO. The Lamefoot Project Area includes portions of Sections 4 and 9, T37N, R33E and a small part of Section 33, T38N, R33E.

2.2 PROJECT AREA AND OWNERSHIP

Echo Bay controls mineral rights on more than 1,000 acres in the Lamefoot vicinity (Figure 7). Only a very small fraction of this area is involved in the proposed commercial production phase. Ownership within the study area is a mosaic of public and private lands, and includes unpatented mining claims on BLM land, fee land owned by Echo Bay, and lands leased to Echo Bay by private entities and by the State of Washington.

A list of all unpatented mining claims within the study area and their BLM recordation numbers are included in Appendix C of this POO.

2.3 PROJECT SCHEDULE

Commercial production is scheduled to commence at the Lamefoot Project upon receipt of permits and completion of the Exploration/Development Phase II program, as described in Section 1.3. Initially, one mining area will be chosen for stoping (commercial production). Others areas will be selected as feasibility allows. It may take several weeks following startup of commercial mining for full production capacity to be realized. Initial stoping is planned to begin in July 1994.

The duration of mining at the Lamefoot deposit will depend upon the results of the exploration/development project currently in progress, and the ultimately feasible production rate sustainable by the resource. Mining rates may also be impacted by other factors such as gold price and alternate ore sources.

Manpower requirements for the commercial production phase are similar to the exploration phase. An increase of approximately 25 people is expected. The exact manpower level will be determined based on the production rate. Currently, thirty-five people are employed at the Lamefoot Mine. The additional personnel required for the commercial production phase may be partially offset by the loss of staff personnel upon closure of the Key Project. The manpower requirement at the mill and administrative site on Fish Hatchery Road should not be affected significantly by the start of the commercial production phase at Lamefoot.

Year-end 1993 reserves in the "proven and probable" and "possible" categories total 2,967,900 → ~~2,966,000~~ ~~2,929,000~~ tons. If all of the probable and possible reserves were deemed minable based on the results of the exploration/development programs, and if a tonnage rate of 1,500 tons per day was assumed, the duration of the uninterrupted commercial mining phase would be about six years. The actual mining rate will be determined by the size and geometry of the resource, as determined by the exploration program. The mining rate is expected to be between 1,200 and 2,000 tons per day. The total waste expected to be produced over the life of the mine, from development of the probable and possible reserves, is 350,000 tons.

The precise scheduling of production on federal land administered by the BLM will also be determined by results of the ongoing programs.

The scheduling of project reclamation is discussed in Section 3.12.

The schedule for production, reclamation and post-closure monitoring is shown on Figure 47.

3.0
DETAILED PROJECT DESCRIPTION

3.1 UNDERGROUND MINING PLAN AND METHODS

Mining of the Lamefoot orebody will be accomplished using underground development and mining techniques.

Underground commercial production at the Lamefoot deposit will consist primarily of two methods of material extraction techniques; access to mining areas and drifting (tunneling) in ore will be accomplished by the use of a jumbo drill and conventional blasting methods. This drill is a rubber-tire mining machine which drills near horizontal holes in a mining face to be blasted. The method results in drifts which can be driven on the level or on an incline, either up or down to an approximate 15 percent grade. Most ongoing development of mining areas during the commercial phase will be accomplished using this method as depicted on Figure 8. Figure 9 shows an oblique view of the completed exploration and development program planned for commercial production.

The majority of the ore commercially extracted within the mining areas will be removed by a long hole, open-stoping technique. This method utilizes drill holes collared in one ore drift (sublevel) and drilled upwards or downwards toward an adjacent sublevel or toward the ore boundary. These holes are loaded with explosives, and the ore between the levels is blasted and falls into a lower level or to the bottom of the active stope, where it is removed. Drilling for blast holes will be done by a single boom production, blast-hole drill, which is an electrically operated machine specifically designed for vertical or angle holes.

Six ore zones have been delineated within the Lamefoot orebody. The locations of Zones 1 through 4 are shown on Figure 10. Zones 5 and 6 are not yet as well defined as Zones 1 through 4. However, available information on Zone 5 indicates that this portion of the orebody is located adjacent and parallel to Zone 4, approximately 100 feet west in the Zone 4 hanging wall. Current estimates indicate that Zone 5 may extend from 27,000 north to 27,400 north between elevations 2,400 and 2,680 feet. Zone 6 is located beneath Zone 4. Current estimates indicate that Zone 6 may extend from 27,000 north to 27,400 north, with a central elevation of approximately 2,200 feet. The mining schedule for all six zones is shown on Figure 11.

3.1.1 Determination of Ultimate Mining Areas

The initial mining through an area of mineralization is accomplished by jumbo mining, as described previously. The limits of ore along this drift are determined by assay of the material removed in conjunction with results of surface and underground exploration drilling previously completed.

Ore limits on a sublevel lateral to a drift are determined by two methods. First, the walls of the drift are tested by coring drill holes perpendicular to the sublevel. This type of core drilling will be done by electric or pneumatic drill rigs. Figure 12 shows a typical layout of holes drilled from a sublevel drift. These closely spaced holes are sampled by lithologic type and assayed. Using these data, the boundary of the ore zone is refined from the configuration determined from surface drilling information.

In selected areas developed during the exploration/development program, short drifts will be driven to the ore boundary to test the reliability of the results of the closely spaced drilling. In the commercial production program, the drill spacing of the lateral core holes will be determined by correlating the results of the short drifts and core holes to the actual ore limits. Currently, the core holes are planned at 25-foot intervals.

Data on ore limits from each sublevel will be evaluated relative to adjacent levels. Based on the sublevel ore boundaries, a three-dimensional model of the ore body is then constructed; and mining areas, or stopes, are designed.

3.1.2 Primary Development

Access to the deposit has been developed starting on the south end at the main portal (elevation 2,600 feet) and extending along the main ramp to a planned elevation of 3,070 feet. Levels developed at nominally 65-foot vertical intervals from this ramp will access the ore zones. The lower limits of the deposit, at its north end, are currently indicated to be at about elevation 2,200 feet.

The planned development program for the main ramp and for level development along the strike of the ore zones is continuing. The program is planned to access ore below elevations of 2,600 feet by developing a ramp at the north end of the deposit.

Additional ramp development is proposed to access narrow ore above the main parts of the ore zones and close to surface, in order to facilitate drift and waste fill mining. Ramp development could also serve as alternate access to the longitudinal stoping areas, if required (i.e., during stope filling).

3.1.3 Development Ground Support

Standard development drifts are 16 feet wide by 15 feet high in waste and 14 feet high in ore. As ground conditions in the Lamefoot Mine are generally very good, a standardized pattern of roof support is used in most development drifts, and is installed shortly after excavation. Roof support consists of 5-foot or 7-foot-long split-set rock bolts and 1-foot by 9-foot steel mats, as shown on Figure 13. On the utilities side of the drift (the side with a ventilation tube, and air and water pipes), 5-foot-long split-sets and 1-foot by 9-foot mats will be installed. In extremely stable ground, no support is required. In less stable ground, increased bolt and mat density, or wire mesh is installed.

3.1.4 Mining Method Determination

Three mining methods are proposed for the Lamefoot deposit:

- Transverse long-hole stoping with pillar recovery
- Longitudinal long-hole stoping
- Drift and waste fill short-hole stoping

The choice of mining method is governed mainly by ground stability considerations and influenced by orebody width, orientation, and depth below the ground surface.

All mined areas will be backfilled as required to ensure no surface disturbance during or after mining operations.

Transverse stoping with pillar recovery is proposed for orebody widths greater than 60 feet. Primary stopes will be backfilled with cemented rockfill to permit safe pillar recovery. The voids resulting from pillar extraction (secondary stoping) will be backfilled with uncemented rockfill. The transverse stoping method is depicted on Figures 14 and 15.

Longitudinal stoping is proposed for most areas with orebody widths less than 60 feet. The stopes will be backfilled with uncemented rockfill. The longitudinal stoping method is depicted on Figure 16.

Level intervals of 65 feet are proposed for both transverse and longitudinal mining. Production equipment will include long-hole drills (LHDs) and haulage trucks. Current rock bolting practice will be supplemented by cable bolting in those areas where stope back spans and rock types dictate.

Drift and waste fill mining is proposed for areas where the orebody geometry is not conducive to long-hole mining, including ore zones of narrow width, and shallow dip and depth below the ground surface. Stope cuts of variable height and width will be advanced and then backfilled with rockfill. Cable bolts may be installed for presupport in cut-and-fill mining areas. The drift and waste-fill stoping method is depicted on Figure 17. The integrity of crown pillars between mined areas and the surface can be improved with the installation of cable bolts.

The stoping and backfilling methods planned for Zones 1 through 4 are shown on Figure 18. Zone 5 will be mined using longitudinal stoping. Transverse stoping methods will be used for Zone 6.

3.1.5 Transverse Stoping with Cemented and Uncemented Rockfill

This stoping method is divided into two phases, illustrated in Figures 14 and 15, and summarized below:

Primary Stoping

- Drive an access drift parallel to the ore approximately 50 feet out into the hanging wall or footwall
- Crosscut through the ore using 16-foot-wide by 14-foot-high drifts on approximately 72-foot centers
- Slab 8 feet off the right rib of each crosscut to attain a 24-foot ore width
- Blast a 6-foot by 6-foot slot raise to the level below at the far end of the crosscut
- Open the slot to full width
- Blast ore into the open slot for the full height in 10-foot to 20-foot increments
- Muck broken ore from the level below with a remote-controlled mucker
- When the stope is empty, backfill to the upper crosscut sill with cemented rockfill
- Repeat the process for the adjacent and overlying primary stopes

Secondary Stoping

- Primary stopes on this level and the level above are completed and filled with cemented rockfill cured at least 30 days
- Drive drilling sublevels 16 feet wide and 14 feet high down the left and right sides of the secondary stope, leaving a 16-foot-wide pillar down the center of the stope
- Install 20-foot cable bolts in the back of both drilling sublevels, using approximately 8-foot by 8-foot spacing
- Connect drilling sublevels at the far end of each stope
- Blast a 6-foot by 6-foot drop raise to level the below
- Expand the slot to the full width of the stope
- Blast ore from both drilling drives into the opening for the full height, using ring patterns in 10-foot to 20-foot increments
- Muck broken ore from the level below using a remote-controlled mucker

- When the stope is empty, backfill to the upper drilling sublevel with waste fill
- Repeat the process for the adjacent and overlying secondary stopes

3.1.6 Longitudinal Stoping with Delayed Waste Fill

This stoping method is illustrated on Figure 16 and summarized below:

- Drive drilling sublevels approximately 65 feet apart along the orebody
- Drill core holes 25 feet apart on both sides of drift to define the ore contact
- Slab sublevels to the full width of the orebody
- Cable bolt the sublevel backs with 20-foot cables on 8-foot by 8-foot spacing
- Blast a 6-foot by 6-foot slot raise from the upper sublevel to the lower sublevel
- Expand the slot to the full stope width
- Blast ore into the open slot for the full height, using ring patterns in 10-foot to 20-foot increments
- Muck broken ore from the lower sublevel with a remote-controlled mucker
- When the stope is empty, backfill to the upper sublevel sill with waste fill
- Repeat the process from the next sublevel above until the top of the ore is reached or the workings are 50 feet from the ground surface

3.1.7 Drift and Waste Fill

This stoping method is illustrated in Figure 17 and is summarized below:

- Drive a drift in ore 14 feet to 18 feet high for the full width of the orebody (12 to 24 feet)

- Backfill to within 2 feet to 3 feet of the back with waste fill
- Repeat previous steps if the ore continues vertically

3.1.8 Stope Stability and Support

3.1.8.1 General

Stope stability and probable support requirements for Zones 2, 3, and 4 of the Lamefoot deposit have been assessed. Several configurations have been considered. Areas of uncertainty have been indicated by suggesting that 50 percent will require support (this may range from 0 to 100 percent but 50 percent is reasonable for estimation purposes).

Stabilities for exposed surfaces, that is, those not supported by backfill, have been rated as follows:

Stable:	Self-supporting, although pattern or spot bolting may be required.
Unstable:	Will require pattern cable-bolt support for stability.
Major collapse:	Will require extensive and heavy pattern cable-bolt support for stability.

Cable bolting will only be required for those levels being used for both production drilling and mucking, with the exception of the 3070 Level of Zone 2, as discussed later. All areas, regardless of cable bolting requirements, will be supported as necessary using the standard development pattern described in Section 3.1.3.

Zones 2 and 4 appear to contain ore widths up to about 130 feet and have been considered for transverse mining. The other zones are generally narrower and the use of longitudinal stoping has been considered. Assessment of stope back and hanging wall stability are summarized below.

3.1.8.2 Zones 2 and 4 Stope Backs

Stability of the backs for these zones has been estimated for stope/pillar strike dimensions of 60 feet and 80 feet, assuming the use of transverse stope/pillar mining, and is summarized below:

60-foot strike	limestone	up to 130 feet wide - stable
	clastics	up to 80 feet wide - stable 80 to 130 feet wide - stable to unstable - 20-foot cable bolts, 6-foot spacing (50%)
80-foot strike	limestone	up to 130 feet wide - stable
	clastics	up to 60 feet wide - stable 60 to 90 feet wide - stable to unstable - 30-foot cable bolts, 8-foot spacing (50%) 90 to 130 feet wide - unstable - 30-foot cable bolts, 8-foot spacing

It is estimated that stope backs with a span of 80 feet will be feasible if additional cable bolting is used in the clastics. For 60-foot spans, cable bolting will likely be required only in some of the wider portions of the orebody. In addition, 30-foot cable bolts on an 8-foot pattern will be used on the 3070 Level for support of the crown pillar. In all cases, cable bolts will be installed from the drill drifts and angled to support the required ground.

3.1.8.3 Zones 2 and 4 Hanging Walls

Stability of the hanging wall has been estimated for both 60-foot single and 130-foot double vertical lifts, and is summarized below. To be conservative, a 60-degree dip has been assumed for the hanging wall.

60-foot strike	limestone	up to 130 feet vertical - stable
	clastics	up to 130 feet vertical - stable
80-foot strike	limestone	up to 130 feet vertical - stable
	clastics	up to 130 feet vertical - stable

In all of these cases, it is estimated that the hanging walls will be stable and not require cable bolting.

3.1.8.4 Zone 3 Stope Backs

Stability of the backs for Zone 3 has been estimated for 40-foot, 60-foot and 80-foot wide spans using strike lengths of 200 feet to 500 feet, assuming the use of longitudinal stopes, and is summarized below.

200 to 500-foot strike	limestone	up to 70-foot span - stable
	clastics	up to 40-foot span - stable 40 to 50-foot span - stable to unstable - 20-foot cable bolts, 6-foot spacing (50%) 50 to 70-foot span - unstable - 20-foot cable bolts, 6-foot spacing

Cable bolt support will likely be required for the clastics in some areas of the backs for spans between 40 feet and 50 feet, and for clastics in all of the backs in excess of 50 feet. Strike lengths will not significantly affect requirements for spans up to 70 feet. Cable bolts will be installed from the drill drifts and angled to support the required ground.

3.1.8.5 Zone 3 Hanging Wall

Stability of the hanging wall for Zone 3 has been estimated for both 65-foot single and 130-foot double vertical lifts, and is summarized below. Hanging wall dips were varied between 60 and 90 degrees for the single lifts. It appears that only the lower two lifts could be taken as a double and these have near-vertical hanging walls.

Single lift	limestone	up to 500-foot strike - stable
(90°H/W)	clastics	up to 500-foot strike - stable
Single lift	limestone	up to 500-foot strike - stable
(60°H/W)	clastics	up to 300-foot strike - stable 300 to 500-foot strike - stable to unstable (see discussion below)

Double lift (90°H/W)	limestone	up to 500-foot strike - stable
	clastics	up to 180-foot strike - stable 180 to 250-foot strike - stable to unstable (see discussion below) 250 to 500-foot strike - unstable (see discussion below)

For single lifts, cable bolt support is indicated for some of the clastic, shallow-dipping areas of the hanging wall if long strike lengths are used (those in excess of 300 feet). A double lift for the lower two lifts of the orebody could need some cable bolt support in the clastics for strike-lengths between 180 feet and 250 feet, and will require cable bolts if opened on strike for over 250 feet.

Cable bolting of hanging walls is unlikely to be practical or cost effective. This leaves two choices - smaller stopes or increased dilution. Single lifts are expected to produce little excess dilution since there are few instances where larger areas of shallow-dipping hanging wall occur in clastics. A double lift for the lower two levels of the zone could result in excessive dilution unless strike lengths are significantly reduced.

3.1.9 Stope Drilling and Blasting Procedures

Production drill-hole configuration is determined by stope width. In narrower stopes, parallel holes will be drilled as shown in Figure 19. In wider stopes, where spans are too great for full width sublevels, ring drilling patterns may be designed.

Vertical and angle blast holes in the stope will vary in diameter from 2¼" to 6" and will be spaced from 5 feet to 15 feet apart. Water may be used during drilling to cool the drill bit and to wash the drilled rock from the hole. This water will be obtained from sumps located throughout the mine. No drilling additives will be used with this water.

The holes will be loaded with an ammonium nitrate-based explosive such as ANFO, water gel, or emulsion. Stick powder (dynamite) or cast boosters will be used. It is anticipated that emulsion will be the primary explosive material utilized based on ground conditions and the dry rock encountered to date. The various rock types respond differently to different types of explosives depending on lithology, degree of fracturing, and alteration. The blasting techniques for given areas will be tailored to accommodate these variables. In addition to different explosive types, different drill-hole sizes, spacing, and patterns influence the effectiveness of the blasting techniques. If emulsions are found to be effective for nitrate control in mine waters, the blasting alternatives will be limited by the use of this type of explosive. However, blasting methods will be continually adjusted according to the variables dictated by lithology, groundwater, and fracturing.

Approximately 1/2 pound of emulsion per ton of ore will be required for mining. This rate translates into a monthly requirement of 25,000 to 30,000 pounds of emulsion to be delivered in two 15,000-pound batches. As much as 40,000 pounds of emulsion may be stored on site at any time. Non-electric blasting caps will be used as detonation devices for the explosives. Care will be taken in the handling of explosive materials to optimize safety of workers and minimize spillage of bulk materials. Water quality monitoring of mine water discharge is discussed in Section 3.13.

Individual slices in the vertical plane of the orebody are blasted into the open stope. The broken ore is accessed below at draw points, as shown on Figure 19.

After blasting and prior to entry, the mining area will be ventilated to exceed air standards prescribed by the Mining Safety and Health Administration. Exhaust air will be vented from the ventilation raise and the main exhaust crosscut shown on Figure 4. The ventilation raise was constructed as part of Phase I of the exploration program. The main exhaust crosscut is an existing adit dating from the early part of the century. Water sprays will be used to mitigate particulate emissions in the exhaust. Depending on final configuration of the mining areas, one or more ventilation raisers or crosscuts may be needed to provide adequate air flow to the working areas. No surface disturbance would occur on federal land associated with these accesses.

3.1.10 Ore and Waste Transfer Procedure

Broken development ore and waste rock will be loaded into low-profile, diesel-powered underground haul trucks at or near the face (end) of the working headings (drifts). Diesel-powered muckers (low-profile front-end loaders) with a 6-yard-rated capacity will be utilized along with 26-ton-rated capacity haul trucks.

The waste rock will be placed in waste fill stopes underground, if available, or will be hauled to the surface and deposited into the waste rock disposal areas shown on Figure 6. Construction design of the disposal areas is discussed in Section 3.4 of the July 1993 POO. Most of the waste rock will ultimately be transported underground and used as backfill material, as described in Section 3.1.11.

Development ore will be hauled to the surface and placed in the temporary ore stockpile area shown in Figure 6. The ore will be sorted in the stockpile area according to source location, this will enable ore from federal land to be weighed and milled separately from ore originating on fee land.

Stope ore will be mucked by remote-controlled, diesel-powered muckers from the bottom level of the stope. This ore will either be directly loaded, or dropped down an ore pass and chute-loaded, into a 26-ton haul truck for transportation outside to the temporary ore stockpile area. Stope and development ore will be sorted at the stockpile area according to source location.

Transportation of ore from the stockpile to the mill is described in Section 3.7.

3.1.11 Backfill Production and Placement

In order to minimize overall surface disturbance on public land and to prevent the possibility of subsidence of the rock overlying the open stopes, backfilling of the void produced during mining will be done following completion of each stope. A summary of the mined and backfilled stopes will be included in the annual report, as described in Section 3.16.

Two sources of rock will be used as backfill. The waste rock disposal areas constructed during exploration and development will comprise approximately 250,000 tons (15.4%) of the backfill material; the remainder will come from the backfill quarry associated with the county road relocation described in Section 3.8.

Two types of backfill that will be placed in the Lamefoot Mine, depending on stope geometry and mine sequencing, are cemented rockfill and waste fill. Cemented rockfill consists of 6-inch minus rock and a Portland cement binder designed to stand to a vertical height of 100 feet. Cemented rockfill formulations, strengths, testing, and quality control are discussed in Section 3.1.12. Waste fill is simply dry, unmodified waste rock used to fill voids, stabilize stope walls and provide a working platform.

The cemented rockfill/waste fill plant is shown in Figure 20 and the production and placement procedures are summarized below:

Waste Fill

- Dump backfill material on surface grizzly
- 24-inch plus material is rejected and transported into the mine for use as uncemented backfill
- 6-inch minus material passes through the lower grizzly and goes to cement rockfill plant
- 24-inch to 6-inch material enters waste fill pass and is transported below ground to 6-foot by 6-foot hydraulically operated incline chute
- Chute material loads ~~Material is chute-loaded into~~ 26-ton haul trucks for transportation to waste fill stope
- Haul trucks ~~end dump~~ End dump material ~~from haul truck~~ into waste fill stope
- When stope is full, blade level with a D-4 or D-6 bulldozer to form working floor for next stope

Cemented Rockfill

- 6-inch minus material from lower surface grizzly enters cemented rockfill pass and is transported underground to cemented rockfill plant
- An approximately 4-foot by 16-foot vibratory feeder supplies material to mixing tower at a rate of 500 tons per hour
- Portland cement binder is transported from the surface storage silo to underground via screw feed to drop pipe to screw feed
- Binder is mixed with water in instantaneous mixer (Figure 21) to 50 percent solids slurry and added simultaneously with cemented rockfill material to mixing tower

- Material and slurry cascades through mixing tower and drops into waiting 26-ton truck
- Material is hauled to cemented rockfill stope and end-dumped
- When stope is full, fill is bladed level with a bulldozer to form floor for stope above

The cemented rockfill is required to have a 28-day compressive strength of at least 400 psi. Mix designs have been evaluated by Hammond, Collier & Wade-Livingston Associates, Inc. (HCWL) and presented in their November 1993 backfill report (HCWL 1993). HCWL evaluated mixes in the laboratory using 2-inch and 4-inch maximum particle sizes for clastic aggregates from the backfill quarry (described as volcanics in the HCWL report), and limestone and clastic aggregates from the underground workings mixed with 3 to 6 percent cement by weight and sufficient freshwater to achieve a water-cement ratio between 0.54 and 0.61. Although some variability in compressive strength was observed in testing, due to maximum particle size, the primary determinants of compressive strength were rock type and cement content. HCWL concluded that the cement content necessary to achieve the required 400 psi compressive strength will be 6 percent for the quarry clastic aggregate, 3 percent for underground limestone aggregate, and 4 percent for underground clastic aggregate. Additional testing is underway on alternative mixes using a water-reducing agent and reduced cement content.

Currently, the preferred cemented rockfill mix is a 6-inch maximum particle size, 4 percent cement, and sufficient freshwater and water-reducing agent to achieve workability without exceeding the water-cement ratio of the batch tests. This mix was selected based on the test results to date, with the understanding that most of the backfill will be either clastic or limestone, and with the expectation that addition of a water-reducing agent will reduce the water-cement ratio and increase compressive strength for a given cement content. Additional testing to evaluate the use of fly ash as a pozzolanic agent and replacement for some of the cement is being considered for further refinement of the mix design. Should fly ash prove to be a viable alternative, geochemical tests will be performed to estimate the leachability of soluble constituents.

The cemented backfill will be sampled and tested as a quality control measure during operation of the cemented backfill plant. Test cylinders will be formed periodically and transported to a laboratory for compression testing. The test frequency will vary depending upon the operating schedule of the cemented backfill plant, but it is expected to be no less than 2 test cylinders per week, as at least one test cylinder is required for each distinct operating period. Additional testing is expected during startup to help establish actual production requirements for achieving the required compressive strength.

Visual evaluation of the stockpiled raw materials, mixing equipment, and finished mix will also be used as a quality control tool. The rockfill materials will be visually evaluated for proper gradation and moisture content. The finished mix will be visually evaluated for proper mixing and placement. Mixing equipment will be recalibrated as necessary and in accordance with manufacturer's requirements.

3.1.12 Subsidence Control and Monitoring

Backfilling of the upper stopes will eliminate the possibility of collapse chimney subsidence (vertical rock fall directly from the roof or ceiling of the mine) breaching the surface. However, minor trough subsidence may develop at the ground surface because stopes will not be completely backfilled during reclamation activities, they will only be backfilled to the extent practicable. Maximum allowable void heights after backfilling within the Lamefoot Mine are shown in Table 4.

Trough (horizontal) subsidence develops over underground openings as the result of the deformation of the rock over and adjacent to the openings (which accompanies redistribution of the loads previously carried by the extracted waste rock and ore). The magnitude of the trough subsidence over underground openings is dependent on the following factors:

- Size of the openings
- Depth of the openings
- Stresses present
- Properties of the rock present

Surface trough subsidence will be small over the planned near-surface Lamefoot stopes, because the elastic modulus of rock is high and the surface crown pillar should be able to support the mining-induced stresses, as indicated in Appendix E (Tables E-4 and E-5). The magnitude of trough subsidence will progressively increase in the case of progressive crown pillar failure, or chimneying, onto underlying backfill.

The backfill program proposed for the upper stopes (2, 3, and 4) is sufficient to eliminate any possibility of a collapse chimney breaching the overlying ground surface through the crown pillars. The crown pillar dimensions are presented in Table 5. The crown pillar thickness shown every hundred feet along strike for the planned upper stopes of the deposit, and the maximum allowable void height, are predicted to prevent chimney collapse from ever breaching the ground surface. As mentioned previously, the maximum allowable void heights are summarized in Table 4.

The effectiveness of the backfilling program in limiting the magnitude of trough subsidence of the surface will depend on the consolidation characteristics and the effectiveness of the modulus actually employed. The results predicted based on conservative estimates of backfill consolidation characteristics are given on Table 6 and show the surface subsidence over: (1) open stopes without crown pillar failure, (2) stopes backfilled with loose uncemented rock fill, and (3) cemented rock-filled stopes. Predicted subsidence varies from 0.95 inches in the 60-foot wide stope to 1.68 inches in the 130-foot wide stope.

The subsidence-induced strains were conservatively predicted by assuming that the entire stope closure was transmitted undiminished through the overlying rock to the ground surface, and by assuming any unfilled opening between the top of the backfill and the bottom of the crown pillar would be transmitted undiminished to the ground surface. Actual stope geometry will be controlled by the cut-off grade determined by the economics at the time, in relation to the ore-grade distribution. The proposed stopes represent current best estimates. The predicted surface subsidence values should not exceed those eventually measured by the planned monitoring program, provided the assumptions required are conservative as intended. Similarly, the calculated ground-surface strains represent local maximum values which should not exceed the strains calculated from changes in the surface monitoring net.

Damage to the ground surface or structures on the ground surface is primarily dependent on the surface strains developed. These strains are the result of flexure (turning or folding) accompanying surface subsidence centered over the stopes. The maximum surface strains are a function of the maximum surface subsidence and the depth. The maximum surface strains were calculated assuming stoping to within 50 feet of the ground surface, and stope widths of 60 feet and 130 feet, using the British National Coal Board (NCB 1975) subsidence prediction method. The results are presented in Table 7, measured in micro-strains.

The predicted maximum strains shown on Table 7 represent the range that would result in the cracking of gypsum plaster, 1000 micro-strain (Voight and Pariseau 1970). As such, the maximum predicted surface strains resulting from trough subsidence should produce nearly undetectable cracking of joints exposed in rock outcrops, should not be observable when soil cover is present, and should minimize surface water infiltration.

3.1.12.1 Surface Monitoring

A surface monitoring program will be carried out using standard surveying techniques. The program will consist of establishing a number of survey stations located above and adjacent to the surface expression of the crown pillar. The positions of these stations in three-dimensional space will then be monitored as mining progresses.

To this end, a series of ~~7~~ 14 level stations will be established over the surface projection of the base of the crown pillar prior to the commencement of production mining. Stations will be located approximately every 100 feet in the dip direction of the orebody, and approximately every 200 feet in the strike direction of the orebody. Proposed station locations are shown on Figure 22. Six ~~Four~~ ~~Three~~ stations are situated over Zone 2, Four ~~two~~ are situated over Zone 3, and Four ~~two~~ over Zone 4. Each station will consist of a concrete pedestal with a permanently attached prism, eliminating the need for a rod man. A typical station is depicted on Figure 23. All stations will be established such that they will be clearly visible from the base station.

The position of the stations will be surveyed on a quarterly basis from a properly established base station located outside of the potential influence of any mining. ~~All station locations will be estab~~

Instruments capable of locating each level station to an accuracy of 0.5 inches will be used. Level surveys will be undertaken quarterly.

The purpose of the monitoring program is to verify that (1) 2 inches of deformation has not developed at the ground surface, and (2) ground movements are not accelerating; occurrence of either of these situations would indicate that the crown pillar is in the process of collapsing, and corrective stabilization measures should be considered. Corrective measures would involve either (1) providing backfill to fill more of the void at or near the top of collapse rubble in the near-surface stope, or (2) preparing to backfill the collapse chimney in the event the ground surface is breached. The indication that the crown pillar is in the process of collapsing would preclude re-entry to the stope for stabilization of the crown pillar and backfilling. Specifically, the Mine Safety and Health Administration (MSHA) would require Echo Bay to prepare and present a re-entry plan for approval. One alternative method of introducing additional backfill into the void is through drillholes angled into the void from the ground surface outside the stope. Another alternative method is to prepare for potential breaches of the overlying ground surface by stockpiling backfill material.

In addition to surface monitoring, regular underground surveys will be carried out during mining to ensure worker safety. These surveys will use systematic observation to monitor ground conditions around mine openings. If conditions are observed to be deteriorating, a surface level survey will be undertaken.

3.1.13 Equipment Requirements/Consumables

Table 8 lists the mobile equipment requirements for the Lamefoot commercial production phase. This listing is based on a production rate of 1,500 tons per day. Different ultimate production rates would result in a slightly different mix of equipment. Some substitutions for comparable items may occur based on availability.

Table 9 lists typical quantities of major materials consumed by the Lamefoot Operation.

3.1.14 Mine Ventilation

Ventilation of the mine will be provided by electric fans. Fans draw air into the portal, and three fans (60 to 100 horsepower [hp] each) will exhaust air from the 6-foot-diameter crosscut or ventilation raise (Figure 4). One 250 hp fan will exhaust air from the exhaust crosscut. Air flow and the routing of air to specific areas underground is facilitated by the use of strategically placed barriers to air flow, and the use of up to six 60 to 125 hp fans to blow air through a ventilation pipe or bag to working areas. As the configuration of the underground workings changes, so does the configuration of the modifications to air flow. The MSHA regulates the air quality in underground workings and requires an updated ventilation design at all times during inspection of the mine. Up to two additional ventilation/escapeways may be required, depending on the exact stope geometry, as determined by the underground exploration program. An updated mine ventilation map will be posted at the mine site office at all times.

To minimize noise from mining operations, a number of mitigation measures have been implemented at the Lamefoot site. The compressor for the mine has been enclosed in an insulated building, and the exhaust from this building is baffled. Fans installed outside of the portal were muffled at the intake and outlet ends, and were moved underground for further mitigation and elimination of their whine. A wall will be placed outside of the exhaust portal to deflect any noise from underground upwards.

Fast-growing poplar trees have been planted on top of the growth medium storage berms to help mitigate noise impacts over the long term. Along with the berm itself, they may be effective in reducing the impact of some site noise, especially in the summer months when leaves are out and when neighboring residences may have open windows.

Should complaints about noise arise as a result of a change in surface activities, noise monitoring will be instituted to characterize the noise and evaluate mitigation measures (Appendix F).

3.1.15 Explosives Storage and Handling

Explosives are stored in an underground locked magazine area. This storage magazine is approved by MSHA. Consumptive use data for explosives are presented in Table 9. All personnel who handle explosives or who are involved in blasting are directly supervised by an employee with a blasting license issued by MSHA. Weekly and monthly safety meetings are held with all operations personnel. Safety in explosives handling is a regular topic of these meetings. The best management procedures for handling explosives are also covered in these meetings so as to prevent spillage at the working face.

3.2 BACKFILL QUARRY OPERATION

3.2.1 Quarry Area and Ownership

The proposed quarry will be located entirely on fee land controlled by Echo Bay Minerals Inc. A portion of the initial access at the north end of the quarry will be located on both public and private land. The quarry is shown on Figure 24.

3.2.2 Summary of Project Activities

The proposed Lamefoot backfill quarry project will consist of mining approximately [illegible] ~~1.4~~ million tons of rock from a single quarry located just to the west of the underground mine. The backfill will be transported to the underground mine via a fill raise and an underground haulage drift that will extend from the underground mine to the floor at the north end of the pit.

stet ok

Briefly, the project will consist of the following:

- Development of initial site access to the top of the ridge at the north end of the quarry
- Logging timber from within the proposed quarry perimeter
- Stripping top soil from within the quarry perimeter and stockpiling the top soil

- Initial development of in-pit haul roads
- Excavation of a backfill raise and cement silo platforms in the bottom of the north end of the quarry
- Concurrent development of an underground haulage drift that will extend from the underground mine to the in-pit fill raise
- Excavation of the fill raise and drilling of the cement distribution boreholes
- Construction of the cement silo and pumping facilities
- Construction of a grizzly over the fill raise
- Mining the quarry and transporting the backfill to the underground mine via the in-pit fill raise
- Relocating the existing county primitive road at the completion of mining
- Removing the backfill grizzly and cement handling facilities
- Reclamation of the site

3.2.3 Project Schedule

The quarry development is scheduled to coincide with commercial development and mining. Work will commence in July 1994 and will continue until approximately the year 2000. Quarry development and extension will begin with clearing and grubbing of the area to be mined. Topsoil will be stripped and stored in the pile to the south of the pit. After final grading, each bench will be topsoiled and seeded before the next bench is completed. Production will proceed in the quarry at a rate approximating 300,000 tons per year, with campaigning in quantities of about 150,000 tons per session.

3.2.4 Geology

The backfill quarry will be excavated in siliclastic rocks that are expected to be similar to the country rocks east of the ore zone. The clastic rocks that will be exposed in the quarry have also been locally intruded by rhyodacite dykes. The intrusive rocks will be mined out and will not be exposed in the pit walls. There is no limestone in the quarry.

3.2.5 Site Topography

The proposed quarry is located within the Okanagan Highland physiographic province of northeast Washington State. The area is characterized by moderate slopes and broad, rounded summits resulting from repeated glaciation.

The quarry is to be excavated into the east-facing slope of a north-northeast-trending ridge that is located immediately to the west of the underground mine.

The elevation of the summit of the ridge varies from 2,700 feet to 2,900 feet at the south and north ends of the quarry, respectively. The inclination of the natural topography of the east slope of the ridge varies from 15 to 50 degrees, and the average slope inclination is approximately 40 degrees.

The floor of the gulley along the east side of the ridge rises from Highway 21 (2,525-foot elevation) to the 2,650-foot elevation at an average inclination of 5 degrees (9%). At this elevation, which is approximately coincident with the south end of the quarry, the inclination of the valley floor decreases to approximately 1.8 degrees (3%). This flat section extends north for approximately 800 feet, forming a relatively flat bench between the 2,650-foot and 2,675-foot elevations. The slope inclination increases to the north of the bench to a maximum of approximately 14 degrees (25%) at the north end of the quarry. The average inclination of the valley floor is 4.25 degrees (7.5%).

3.2.6 Mining Operations

3.2.6.1 Backfill Requirements

The projected amounts of cemented and uncemented rockfill that will be required on yearly basis ~~is~~ are presented below:

Year	Cemented (tons)	Uncemented (tons)	
1994	72,000	80,000	
1995	73,000	234,000	
1996	42,000	261,000	
1997	63,000	244,000	
1998	58,000	244,000	
1999	37,000	195,000	
2000	0	10,000	
TOTAL	345,000	1,268,000	1,613,000 tons

Approximately 220,000 tons of the rockfill will be obtained from development waste rock. The remaining 1,393,000 tons will come from the backfill quarry.

The backfill requirements listed above are based on 1993 year-end reserves. As the reserve estimates are further refined, backfill requirements may change. The ultimate capacity of the backfill quarry as it is shown in Figure 24 is approximately 2,800,000 tons ~~(Figure 24)~~, which allows for possible variation in in-situ and backfill unit weights, ~~and~~ for variation in ore and waste rock tonnages, and for possible aggregate requirements ~~as of road maintenance by~~ Ferry County for future road maintenance needs. (Marshall, 1994). Backfill material will be obtained on an as-needed basis; therefore, no quarry material will be left over or stockpiled for significant periods of time.

3.2.6.2 Acid Generation Control

The rock within the quarry contains pyrite, as well as oxides and hydroxides of iron resulting from the weathering of pyrite. Consequently, the limits of the quarry have been selected so as to minimize the acid-generating potential of the backfill that is to be placed in the underground mine. A total of 11 reverse-circulation holes have been drilled in the quarry for the purpose of obtaining samples to test for acid generation potential. A block model has been developed of the acid-generating and acid-neutralizing potential of the rock consigned to the underground mine, and this model has been used to ensure that the ratio of these potentials will be greater than 3. The addition of cement to the rockfill will further reduce the potential for acid generation in the underground openings. See Section 3.15 for details on the Geochemical Monitoring Plan. Appendix G provides a summary of water quality data obtained to date.

3.2.6.3 Quarry Configuration

The extent and shape of the quarry have been determined on the basis of:

- Required tonnage (approximately 1.6 million tons)
- Acid generation control
- Minimum bench mining widths of 75 feet
- Haul road locations with maximum grades of 12 percent
- Location of the fill raise and the cement handling facilities

The quarry will be approximately 1,700 feet long, and the maximum width will be approximately 600 feet. The maximum wall height will be approximately 125 feet. ~~All highwalls over approximately 5 feet in height will have a set-back distance from the county road of at least 50 feet.~~ Typical cross sections of the quarry are shown on Figures 25 and 26. Initially, the fill raise and cement silo platform described in Section 3.1.11 will be excavated in the northeast co~~m~~rner of the quarry so that backfill can be placed underground as soon as possible.

The bench configuration of the cut walls in the quarry is dependent upon several parameters, including bench heights, bench face angles, berm widths, and overall slope angles. The quarry will be excavated in 25-foot-high lifts in order to enhance fragmentation during blasting and to enhance safety during excavating. The walls above the fill raise platform at the north end of the mine will be excavated at overall slope angles of 45 degrees, and a 16-foot-wide catch-berm will be left at the toe of every bench. The remaining walls will be excavated at slope angles of 37 degrees, and the catch-berms will be 24 feet wide on every bench. Relatively shallow overall slope angles have been chosen so as to enhance the long-term stability and safety of the slopes. Rock mechanics investigations in support of the proposed slope designs are presented in the Appendix of the Backfill Quarry Report (Golder Associates 1994).

3.2.6.4 Haul Roads

The in-pit and out-of-pit haul roads have been designed to accommodate two-lane traffic. The anticipated maximum vehicle width is approximately 10 feet, and running surface widths have been designed at 30 feet, three times the maximum vehicle width. Overall road widths have been designed at 40 feet to allow for the placement of berms and ditches along the outside of the haul roads. Haul road grades vary from 4 to 12 percent.

Dust will be minimized by sprinkling water or a DOE-approved chemical binder on the haul roads.

3.2.6.5 Mining Procedures and Equipment

The quarry will be excavated by conventional open-pit mining methods involving drilling and blasting, loading, and hauling.

Access to the site will be gained by the existing primitive road that runs northeast from Highway 21. Initially, the upper benches of the quarry will be accessed from exploration roads that are located on land within Echo Bay's unpatented mining claims and on private land. These roads will be minimally upgraded to accommodate excavation equipment, and will be used to access the upper benches from approximately the 2,875-foot elevation down to approximately the 2,825-foot elevation. An interim in-pit haul road located within the fee land will be pioneered with air-track blasthole drills and dozers. This haul road will be used to access the quarry down to approximately the 2,775-foot elevation. The remainder of the quarry will be accessed from a permanent haul road that will be excavated along the east edge of the pit floor.

Production blastholes will likely be drilled with boom-mounted percussion drills. Front-end loaders will be used to load the fragmented rock into 20-ton to 30-ton haul trucks. The rock will be hauled and dumped into an underground fill raise. It is expected that the desired degree of rock fragmentation will be achieved during blasting, and that secondary crushing will not be necessary. The blasting procedures are discussed in further detail in the Appendix of the Backfill Quarry Report (Golder Associates 1994).

The quarry configuration has been designed on the assumption that the following major mining equipment, or their equivalents, will be used:

Front-End Loader (CAT 988 or equivalent)	1
Haul Trucks (CAT D250 25-ton articulated truck or equivalent)	2
Dozer (D-9 or equivalent)	1
Blasthole Drill (Cable of drilling 30-foot-deep blastholes)	~~2~~ 1
Equipment Service Vehicle	1
Water Truck	1
Supervisory and Crew Transport Vehicles (Pick-up trucks)	2

The above equipment list may be revised to account for variations in mining rates and conditions, and underground backfill requirements.

To minimize noise impacts from the front-end loader, which has a high noise-level backup alarm (as required by MSHA), the alarm will be replaced by strobes during nighttime operations.

3.2.6.6 Workforce

At present, it is uncertain whether the quarry will be mined by Echo Bay personnel or by a contractor.

3.2.7 Reclamation

It is Echo Bay's intent to reclaim the pit and haul roads following the completion of mining operations. The various aspects of the reclamation plan are addressed in Section 3.12.

3.3 FIRE-FIGHTING EQUIPMENT/EMERGENCY RESPONSE

All light vehicles on site will be equipped with an axe, shovel, bucket, and fire extinguisher during fire season. In addition, all vehicles and other internal combustion engines will have adequate spark arresters. Telephones are located at the core shed and office building on the private land within the study area. These locations are within ½ mile of all of the proposed surface work sites.

Phones are also located underground for communication with the surface. All underground personnel are aware of the location of these phones and are informed of emergency escapeway locations. A map showing emergency escape routes is posted at the sites, and routes are clearly marked underground.

Echo Bay will promptly comply with any emergency directives by BLM or the State of Washington, and will obey any fire precautions imposed on operations during the summer fire season. During periods of high fire danger (determined by the BLM or state), the water trucks servicing any surface drill rig will have pumping capabilities and a minimum of 100 feet of hose. The use of existing water sources on site will also be available to fire-fighting efforts in the area.

An ambulance will be present on site at the Lamefoot Mine during production. This vehicle and the ambulance at the mill site, along with trained emergency medical technicians employed by Echo Bay, will be available to respond to emergencies at the mine.

A trained mine rescue team is on call at all times to respond to underground mine emergencies. Trained emergency medical technicians are members of this team.

3.4 POWER/WATER REQUIREMENTS

Electrical power is purchased from the Public Utilities District of Ferry County. Current power consumption is about 11,000 - 20,000 KW hrs/day. During production, the requirements will be about 20,000 - 25,000 KW hr/day. A substation at the site reduces the power from the Public Utilities District distribution line to 4.16 KV. A smaller substation at the portal further reduces voltage to 480 V.

Water requirements for Lamefoot and other elements of the Kettle River operations are shown on Table 10.

3.5 TECHNICAL STUDIES

3.5.1 Rock Mechanics

A geotechnical assessment of all of the stoping areas was completed to project final backfilling requirements for the individual stopes. This study will continue throughout the mine life. Design criteria for the stopes and for backfilling will be based on detailed geotechnical information obtained in the mine workings. This information is acquired by mapping of fractures with particular emphasis placed on logging of orientation, spacing, planarity, roughness, alteration, and continuity. Ultimately, design of backfill requirements will be based on a zero subsidence objective.

These data will be used in conjunction with measurements of compressive strengths and coaxial and triaxial shear strengths of the host rock, in order to model the geotechnical characteristics of the ore body. The results of a computer simulation presented in Appendix E identifies possible areas of long-term instability for design of backfill requirements.

Monitoring sites will be established on the surface over mining areas to assess if any small amounts of ground movement are occurring as a precursor to subsidence. These permanent monitoring sites will be checked on a schedule determined by mine engineers that is based on the mining areas being used. Stations will also be established underground for similar studies of ground movement. Discussion of the surface monitoring program is presented in Section 3.1.12.1.

3.5.2 Geochemistry of Underground Mine and Backfill Quarry

Echo Bay has commissioned several studies on this topic during the period between 1990 and 1994. Results of that testing and its environmental impact implications are discussed in detail in the FEIS (USFS 1992; Sections 3.3, 4.3, and 4.16.3).

3.6 WASTE ROCK DISPOSAL AREAS

The waste rock disposal areas shown on Figure 6 currently contain approximately 260,000 tons of waste rock material, which is primarily limestone. Total projected waste-rock production is estimated to be 350,000 tons, assuming that no additional mineralization is located and that currently indicated mineralization is entirely mined. Approximately 250,000 tons of the waste rock will be used as either cemented or uncemented backfill in the workings. The remaining 100,000 tons will be recontoured and reclaimed in place around the portal area.

Approximately half of the waste rock to be used as backfill will be obtained from the workings and hauled directly to the backfill area without storage on the surface; the other half will come from the waste rock disposal areas near the main portal. The 160,000 tons of waste rock from the surface will be moved underground in two phases. The first phase will involve approximately 60,000 tons at the start of commercial operations and during initial development of the backfill quarry and appurtenant facilities. The second phase will involve approximately 100,000 tons near the end of commercial operations in preparation for closure.

The waste rock disposal areas are and will continue to be constructed according to strict materials-handling criteria. Clastic rocks will be identified underground and will be handled separately from limestone and low-grade ore. Clastic rocks have been encapsulated in the waste rock piles and surrounded by acid-neutralizing material (see Figure 6).

Most of the waste rock mined during the exploration/development program at Lamefoot is composed of highly neutralizing limestone. Waste rock which contains sulfides in concentrations of greater than 4 percent, is segregated in the disposal area for subsequent encapsulation by neutralizing rock. A detailed discussion of the determination of rocks with acid generation potential, and the handling and composition of rocks is found in the Key Project FEIS (USFS 1992).

At Lamefoot, development drifting has been designed specifically to occur in limestone, to the extent possible, because it is a highly neutralizing rock type. Only a very small percentage of the development waste may be potentially acid generating (less than 8% during preproduction phase). Total waste rock tonnage is estimated to be 350,000 tons during the mine life, assuming that no additional mineralization is located and that all currently indicated mineralization is mined. Potential acid-rock mitigation alternatives for both the underground workings and surface waste rock disposal areas are presented in the Lamefoot rock characterization report (Appendix H).

Ore and Waste Rock - Following mine closure, sediments from underneath areas where ore and waste rock piles have been stored on the land surface will be sampled to detect for possible metal, acid, or sulfate contamination to surficial materials.

3.7 ORE TRANSPORTATION

The ore will be temporarily stockpiled at the Lamefoot site and then loaded into highway haul trucks by a front-end loader for transportation to the mill. These trucks will probably be rated at a 20-ton capacity and will trail a 10-ton or 20-ton "pup" or tandem trailer. All trucks will conform to requirements of the Washington State Department of Transportation and the County Road Department, and will be permitted by the agencies if and as required.

~~An alternate method of ore loading into haul trucks may be utilized. This method would require unloading by underground trucks into an enclosed crusher assembly where the rock would be reduced in size and batched by conveyor into highway trucks.~~

The primary route for the haul trucks will be south along the two-lane State Highway 21 to the county Cooke Mountain Road then east to the Jack Mays Pride Road. The ore will be hauled on Jack Mays Pride Road to Fish Hatchery Road, then east to the Kettle River Mill (Figure 2). A one-way trip for the ore haul includes a total of 5.0

miles of state highway and 3.8 miles of county road. This route has been developed in cooperation with the County Commissioners, based on considerations expressed by the County Engineer and Director of the Public Works Department, as well as public input. Should the county or the state request that other routes be used, Echo Bay will abide by those requests.

During commercial production, ore hauling will range from 30 to 70 round trips per day, depending on the truck size and actual production rate at a given time. Improvements of the county Cooke Mountain Road, to be completed in 1993-94, will allow for year-round (all-weather) usage. Echo Bay and Ferry County have entered into an agreement whereby Echo Bay has reimbursed the county for certain maintenance costs of that road. Further, Echo Bay will contribute funding to the capital costs of upgrading the road itself.

The ore haul will be restricted to 12 hours per day between the hours of 6:30 a.m. to 6:30 p.m., seven days per week.

Periods of weight restrictions and/or construction on State Highway 21 would change the description of haul traffic. Prior to weight restrictions going into effect, ore hauling may increase in order to stockpile ore at the mill. Haul traffic may exceed the 60 to 140 Average Daily Traffic (ADT) range occurring under normal operating conditions. During periods of road construction, which are scheduled to occur in 1995, some hauling may take place at night in order to avoid traffic delays. Hauling would remain within the 60 to 140 ADT range during these periods.

Public access to the Key Mill is also provided via Fish Hatchery Road. This route is not proposed for mine-related traffic, but is used by employees going to and from work. To mitigate impacts from transportation, Echo Bay has committed to the following:

- Haul truck drivers will be required to follow all posted speed limits and to adhere to any special limitations on speed or weight imposed by the state and county in order to mitigate damage to road surfaces.

- Echo Bay has agreed to contribute funding to the county's upgrading of the Cooke Mountain Road so that an all-weather surface can be constructed. Echo Bay has also agreed to enter into a road maintenance agreement to provide funding for the normal repairs of all county roads along the haul route.

- By obeying traffic laws, haul truck drivers can reduce the chances of vehicle accidents. Drivers reported to be speeding or driving recklessly will be subject to disciplinary action or dismissal.

- Haul trucks will be supplied with wildlife whistles to decrease the likelihood of hitting deer on the haul route.

- All trucks will be equipped with mufflers in good operating condition so that engine noise is kept to a minimum.

- Trucks will be kept in good mechanical operating condition so that emissions are minimized.

- Echo Bay proposed to continue to use the Cooke Mountain Road in order to mitigate concerns of residents on Fish Hatchery Road relating to dust, noise, and traffic. Echo Bay also applies dust suppressants at intervals near residences along Fish Hatchery Road to decrease dust in those areas.

- A sign will be placed at the west end of Fish Hatchery Road restricting through truck traffic. This should decrease inadvertent use of this road by vendors or contractors unfamiliar with the area or with Echo Bay's policy of using the Cooke Mountain Road.

- Signs have also been placed on the entrance to the relocated Wolfe Camp Road so that the mine entrance to the Lamefoot facility is clearly marked. This will prevent unnecessary mixing of public traffic and mine traffic on Wolfe Camp Road.

3.8 COUNTY ROAD RELOCATION

The primary access to the Lamefoot site is the previous alignment of Wolfe Camp Road (Figure 6), which was vacated by the county on January 11, 1993. The relocated Wolfe Camp Road provides access to residences to the north.

During 1991 and 1992, surface exploration drilling was conducted from this road, and from roads to the east of and above the county road. During construction on one of these drill roads, significant amounts of blasted rock were deposited downslope, forming a talus slope above Wolfe Camp Road. In order to stabilize this rock, Echo Bay dislodged as much as possible, and netted the residual talus with chain link. This temporary stabilization eliminated a potentially hazardous condition along Wolfe Camp Road.

To avoid long-term instability present in the area, Echo Bay plans to relocate Wolfe Camp Road to the west side of the small valley which the county road occupies. This relocated alignment would largely occupy a previously disturbed area in a quarry (Figure 24). Some new surface disturbance (7.4 acres) would occur on private land as a result of this relocation. Permanent right of way would be granted to Ferry County by Echo Bay upon completion of the new alignment of Wolfe Camp Road. All disturbance associated with the relocation will occur on fee land owned by Echo Bay.

In order to provide for safe and stable rock slopes on the roadcuts above the new Wolfe Camp Road, the oversteepened slopes of the existing quarry area would be laid back to slopes at a safe effective angle of 37 degrees. Ultimate maximum slope angle will meet the requirements specified by the County Engineer. Ferry County will regulate the design and construction of the new road alignment on private land. The material from the excavation is planned for use as supplemental backfill in the open stopes of the Lamefoot Mine.

The Ferry County Road Department will design the intermediate and ultimate road configurations, using county standards. Echo Bay will reimburse the county for engineering costs associated with this project. The county will also determine the traffic flow along the county road. Conceptual designs for the road are shown in Figures 24a, 24b, 24c, and 24d.

Blasting in the excavated area will occur on a predetermined schedule designed to occur during lowest traffic volume periods on Wolfe Camp Road, but limited to daylight hours between 6:00 am and 6:00 pm. This schedule will be coordinated with the county and the sheriff's department. Delays caused by blasting on Wolfe Camp Road will be minimized and should not exceed 15 minutes. Blasting is not planned to occur more than once per week.

3.9 NOXIOUS WEED CONTROL

Ferry County Mining Ordinance General Provision 3.02.1 states "All exploration operations will comply with the requirements to control noxious weed infestation in accordance with the Rules and Regulations of the Ferry County Weed Control District."

Noxious weeds will be controlled throughout the life of the operation. Revegetated areas will be qualitatively evaluated on an annual basis to assess weed populations. Weed control on private and BLM land is accomplished by promoting vegetation growth and by spraying selected herbicides, as recommended by the Ferry County Weed Board. Spraying will be used primarily for the control of knapweed; however, the area will be monitored for all noxious weeds, and herbicide application will be designed to meet the needs of the Ferry County Weed Board. Spraying on federal land will conform to conditions specified by BLM.

Heavy equipment will be thoroughly cleaned before movement into reclaimed areas, in order to prevent the spread of noxious weeds. This will reduce the seed source available to invade reclaimed areas. Monitoring and control of weeds on reclaimed areas is especially important in effectively limiting weed populations.

Details for controlling noxious weeds are included in Section 3.12.10.

3.10 LAMEFOOT EXPLORATION

An extensive mineral exploration, surface drilling program has sampled the most promising ground in the immediate vicinity of the Lamefoot deposit. However, it is possible that additional surface exploration drilling may be warranted within the area (Figure 7). Also, additional (in-fill) drilling of shallower portions of the previously defined deposit may be required to better delineate ore boundaries or to study geotechnical characteristics of the ore and host rock.

If in-fill drilling is required, no additional drill roads or other incremental surface disturbance will be needed. However, for areas not previously explored, new drill pads will be constructed for access by the drill rigs. These roads will be constructed and reclaimed by methods discussed in the Lamefoot POO (Echo Bay 1992a), the EA (BLM 1992a), and in Section 3.12. To summarize, the drilling program would involve drill holes as well as the associated road sites. The exact location of these roads will be determined by the location of the target to be drilled. These locations will be established based on results of the exploration and production phases.

The drill holes could include both 6-inch-diameter, circulation rotary drill holes and 3 to 4-inch-diameter coreholes. The depth of the holes could range between 250 feet and 1,000 feet. Each site will be flagged prior to drilling.

Water will be supplied by 2,000-gallon water trucks and distributed as necessary. Sumps may be necessary at some of the drill sites in order to collect drilling fluids and to minimize overflow onto the surrounding landscapes. The sumps will have a sloped ramp so that animals can climb out should they venture into the sump. After drilling is completed at each site, the drilling fluids will be allowed to evaporate and infiltrate, and the sump will be backfilled as soon as possible.

Drilling may include the use of both truck and skid-mounted core rigs, as well as truck and track-mounted rotary rigs. New drill site disturbance will be minimized to the greatest extent possible. This includes limiting surface disturbance drill-pad size to 30-feet by 30-feet, locating drill sites on roads or in natural clearings wherever possible, and drilling multiple holes from each pad, where feasible.

Road construction and reclamation on steep slopes may be difficult. Exploration of these areas will occur through helicopter transport of drill rigs or through hand-pulling of the rigs to the drill sites. Drill pads for these areas will be 20-feet by 20-feet, unless additional cleared area is absolutely necessary in order to ensure the safety of helicopter loading/unloading. The maximum area that would be cleared at any of the helicopter sites would be 50-feet by 50-feet.

None of the existing drill pads or the proposed pads will be contiguous, and all drill pads will be separated from each other by zones of undisturbed vegetation.

Drill Roads

Existing trails and clearings would be preferentially used for providing access to drill pad sites, with little or no new construction requiring heavy equipment. In areas requiring heavy equipment, a D-8, D-7, or equivalent bulldozer will be used.

Road and trail widths may vary somewhat, depending on the intended use of the road and on the terrain (that is, a road built in relatively steep terrain will result in more disturbance than one built on the flat). In general, road widths would range from ~~20~~ 15 to ~~30~~ 20 feet, depending on topography, but will be minimized where possible. Water bars and safety berms/ditches will be installed to disperse runoff water along constructed roads.

Reclamation of any new roads and drill pads would be carried out as described in Section 3.12.

3.11 SURFACE DISTURBANCE

Current surface disturbance at Lamefoot is related to the existing site facilities supporting the underground exploration and the drill roads used for surface exploration. No surface disturbance on BLM land is anticipated to result from the underground exploration or mining, but up to 5 acres of additional surface disturbance could occur with further exploration.

The acreage of additional surface disturbance resulting from surface exploration on BLM land will be difficult to assess until specific exploration targets have been identified and the programs planned. However, any additional disturbance on federal land is expected to be less than 5 acres during the life of the project. This disturbance would be similar in nature to existing disturbance described in the POO (Echo Bay 1992a) and the EA (BLM 1992a). Reclamation would be conducted as described in the two previously mentioned documents and in Section 3.12.

3.12 RECLAMATION

3.12.1 Introduction

This plan presents reclamation activities for final mine closure and summarizes the reclamation activities previously approved by the BLM for the exploration drill pads. BLM lands disturbed by project activities constitute a small portion of the total disturbance. Most surface and subsurface disturbance is located on private land controlled by Echo Bay. This reclamation plan has been prepared to describe the reclamation of the BLM lands. Reclamation plans for private lands are also being provided for completeness.

Echo Bay will reclaim disturbance where conditions and current reclamation technology permit, and as required by BLM, state, and county regulations. Historical and current land uses include livestock grazing, wildlife habitat, forest management, and mineral extraction. The goal of reclamation is to return the federal land at the site to a post-mining condition which is consistent with the preferred BLM alternatives for forest, wildlife, and recreation management (BLM 1985, BLM 1994).

Reclamation plans with regard to mining activities on BLM administered lands are similar to those already approved and successfully used by Echo Bay in past activities, and are discussed below as they pertain to road building, concurrent reclamation, interim reclamation, and final reclamation of the disturbed areas. Reclamation plans presented here are, to the extent applicable and appropriate, based on guidelines contained in the BLM Reclamation Handbook (BLM 1992b).

Echo Bay's proposed post-mining land use is for wildlife habitat. The reclamation species mix will return the surface facilities to the predominant bluebunch wheatgrass (*Agropyron spicatum*)/Idaho fescue (*Festuca idahoensis*) grassland community which the site supported prior to disturbance. The species proposed will provide initial site stabilization to prevent erosive soil loss and provide browse for wildlife. The mosaic of open grassland and timbered stands will be well suited to provide the diversity of cover, edge, and browse which provide deer with winter range.

3.12.2 Reclamation Schedule

Reclamation activities will be scheduled to occur as soon as practical after the mining activities are completed, thus minimizing erosion and sedimentation problems (see Figure 46). In general, reclamation activities will be timed to take advantage of optimal climatic conditions. Final grading, drainage, and sediment control establishment will occur over the late spring and summer months. Seedbeds will be prepared in late summer or early fall, just prior to seeding. Seeding will be completed in mid to late fall in order to take advantage of winter and spring moisture. If seeding is not completed prior to the onset of winter, or if fall seeding is unsuccessful, early spring seeding could occur as an alternative.

Many of the reclamation activities can not occur until near the time of final mine closure. Areas such as underground workings and surface facilities associated with exploration will remain active until mine closure. However, during the anticipated life of the project, interim and concurrent reclamation will occur to reduce erosion and the potential for off-site degradation.

Interim reclamation refers to reclamation efforts on lands disturbed during the course of a project that, while not at final reclamation contours, will not be redisturbed for a significant time period and, therefore, require interim stabilization. To reduce erosion and sedimentation during the life of the operation, disturbed areas will be temporarily revegetated. For all areas subject to interim reclamation requirements, slopes will be reduced as much as feasible; the surface will be appropriately prepared/tilled to receive broadcast seed, starter fertilizer (16-48-37) will be applied, and the seeded areas will be dragged to facilitate burial. These measures will minimize the potential for accelerated erosion and will optimize conditions for successful establishment of the interim vegetative cover. Straw mulch will be applied, as required, at a rate of 2 tons per acre on slopes in excess of 15 percent but less than 30 percent. The straw will be crimped into the soil surface on the contour. On slopes in excess of 30 percent, straw mulch will be applied at a rate of 1.5 tons per acre and sprayed with an agency-approved tackifier; netting will be applied and stapled over straw mulch for specific areas of concern such as highly erodible soils placed on steep slopes in excess of 50 percent. Salvaged coversoil will not be used in interim revegetation unless readily available. On private lands, interim reclamation areas will be broadcast-seeded at rates of 25 to 40 pounds pure live seed (PLS) per acre, with an interim seed mixture consisting of the following:

•	60% Luna pubescent wheatgrass	(*Agropyron trichophorum* var. Luna)
•	30% Sherman big bluegrass	(*Poa ampla* var. Sherman)
•	5% alfalfa	(*Medicago sativa*)
•	5% white Dutch clover	(*Trifolium repens*)

This seed mix was recommended by U.S. Soil Conservation Service personnel. The higher rate will be used on difficult sites, such as on the face above the adit entrance. Seed, fertilizer, and mulch will again be added on both public and private lands if initial seeding is unsuccessful. The cover soil stockpiles will require interim reclamation.

On federal lands, interim reclamation areas will be broadcast-seeded at rates of 25 to 40 pounds PLS per acre, with an interim seed mixture consisting of the following native species:

- 30% mountain brome (*Bromus marginatus*)
- 30% slender wheatgrass (*Elymus trachycaulus*)
- 20% Idaho fescue (*Festuca idahoensis*)
- 20% bluebunch wheatgrass (*Agropyron spicata*)

Concurrent reclamation refers to reclamation activities which can be carried on at the same time as ongoing mining activities. Concurrent reclamation can be advantageously employed on disturbed areas that have served their purpose and are ready to be graded to final reclamation contours. Such areas will include disturbance associated with the

Wolfe Camp Road relocation and diversion ditches. Drill pads and access roads may be reclaimed concurrently if it is determined that they are not needed for future activities. Where possible, during the life of the Lamefoot Project, disturbed lands will be reclaimed concurrently with ongoing mining operations, using final reclamation procedures.

Most reclamation activities will take place at the time of mine closure and will be considered final reclamation. The areas to undergo reclamation at mine closure will include waste rock disposal areas, the underground workings and associated surface facilities, and any drill pads and access roads which will be needed until mining ceases. Final reclamation will be implemented upon the completion of mining and exploration; these procedures are discussed below in detail.

3.12.3 General Reclamation Procedures

This section includes the general steps to be followed in reclaiming each of the disturbance areas.

3.12.3.1 Coversoil

The use of suitable coversoil is a critical component of the revegetation plan and will enhance the recovery of reclaimed mine facilities. Coversoil is defined as the soil of the A and B horizons or referred to as all suitable solum materials that will sustain plant growth. Coversoil will be used to facilitate revegetation of areas disturbed by the mining operation.

3.12.3.2 Fertilization

Disturbance areas associated with exploration at Lamefoot and other areas of the Kettle River Project have been revegetated successfully without fertilization. Based on these results, it is anticipated that fertilization may not be necessary. Native species are better adapted to low levels of available nutrients. However, fertilizer at a rate of 50 lbs N/acre, 100 lbs P/acre, and 60 lbs K/acre will be applied to areas where nutrient deficiencies are observed in the re-established vegetation. Fertilizer will also be applied to critical revegetation areas such as slopes in excess of 50 percent. Care will be taken immediately after any necessary fertilization efforts to ensure existing erosion control features/measures are in place and prepared to limit runoff into springs and wetlands during a storm event (Section 3.12.3.4).

Samples will be taken from each mine facility (waste rock disposal areas, surface facilities, drill pads and exploration roads, and coversoil stockpiles) to characterize baseline physical and chemical plant growth conditions prior to reclamation activities. If nutrient deficiencies are indicated by revegetation monitoring, Echo Bay will add fertilizer to facilitate the establishment of seeded species. General fertilization guidelines (Soltanpour et al. 1985) indicate that plant growth material containing less than 6 ppm N, 10 ppm P, and 60 ppm K will require supplemental fertilizer for plant growth. A typical fertilization program will include approximately 50 lbs N/acre, 100 lbs P/acre, and 60 lbs K/acre if soil analysis indicates deficiencies. If fertilizer is required, care will be taken to avoid the establishment of a plant community which is dependent on high nutrient levels.

3.12.3.3 Cultural Treatments

Cultural treatments typically refer to soil-modification practices that create more favorable conditions to facilitate plant growth by: (1) initiating and maintaining a stable soil system, (2) reducing erosion of surface soils, (3) increasing soil moisture and reducing evaporative losses, (4) extending the season of seeding by moderating local microclimates, and (5) modifying microenvironments to create a more diverse plant community. Typical cultural treatments that can be used to facilitate plant community development include soil ripping, tilling, harrowing, seedbed preparation, mulching, and erosion control measures. All mine units which are compacted (i.e., roads, level surfaces on waste rock disposal areas, and surface facilities) will be deep-ripped with bulldozer-mounted ripper bars or a chisel plow to a depth of 12 to 18 inches, in order to loosen the plant-rooting zone, create an adhesive surface for the coversoil application, and incorporate fertilizer materials. Deep ripping will increase infiltration by decreasing the bulk density, thus reducing runoff and erosion from the reclaimed slopes. Scarification (shallow ripping to 8 to 10 inches) will be conducted on mine units which do not have deep subsoil compaction. Ripping on any reclaimed slopes will occur parallel to the contours, where possible. On slopes with subsoil compaction that are too steep to be worked on the contour, ripping will occur perpendicular to the contours, and a chain drag will be attached behind the ripper to eliminate furrows and to reduce erosion due to channelized flow. Coversoil application on slopes will result in a loose soil surface that is a receptive seed germination environment.

3.12.3.4 Erosion and Sediment Control

Erosion control will be accomplished by the two existing diversion channels which collect runoff from upgradient of the surface facilities. The diversion ditch channels flow to two infiltration ponds located in the southern and northeastern parts of the surface facilities. Sedimentation ponds will be monitored at least once per month during the summer months, weekly during spring snowmelt, and following large precipitation events. Sedimentation ponds will be cleaned, as necessary, using a backhoe. Sediment removed from the ponds will be added to the coversoil stockpiles. Diversion ditches and infiltration ponds will be backfilled with sidecast or materials excavated from the ditches or ponds during construction.

Sediment control structures will be placed in ditches and below unrevegetated slopes to aid in erosion and sediment control, as necessary. Hay bales, silt fences, matting, and other sediment management practices will be used to aid in erosion and sediment control. Only hay that is free of noxious weeds will be used, reducing the potential for establishment of noxious weeds. Erosion and sediment control measures also include soil handling and grading techniques to enhance stability and reduce sedimentation, and revegetation practices to provide soil-stabilizing vegetation cover.

Water handling measures to control runoff and sedimentation during operations, and until revegetation has been successfully established, include:

- The disturbed area will be kept to a minimum at any given time through interim and concurrent reclamation.

- Drainage structures constructed on access and haul roads will include properly installed channels, water bars, cross drains, culverts, sediment traps, and silt fencing.

- Rapidly developing species are included in the seed mixture to provide rapid stabilization.

- Topsoil redistribution and revegetation will occur in the first appropriate season after cessation of mining.

- Mulch (and tackifiers on hydroseeded areas) will be applied to aid in erosion control and moisture retention, as necessary. It is anticipated that mulch will be applied to topsoil stockpiles and to slopes steeper than 15 percent.

- Revegetated areas will be protected from disturbance by placing signs and barriers to restrict traffic until vegetation is established.

- Interim revegetation will be used to stabilize coversoil stockpiles.

Grading during reclamation will be designed and conducted to minimize the potential for erosion, and measures include:

- Reclaimed slopes will be inspected periodically. Any rills and gullies that develop will be stabilized and revegetated by backfilling, hand compacting, and seeding with the interim seed mixture.

- Fill slopes and other potential sediment sources will be visually inspected throughout the operation to allow early detection of erosion and vegetation problems. During critical runoff periods, such as the spring snowmelt, inspection of some layer fills and erodible areas will be on a more frequent basis.

- Road grades have been designed such that natural drainage patterns are disrupted as little as possible.

3.12.3.5 Grading and Stabilization

Slopes will be shaped for reclamation upon completion of the active life of each project component. Depending on the type of material, erodibility, and the practical considerations of the mining process, overall slope grades will vary. Grading will be accomplished using bulldozers. Figure 27 presents the proposed final topography following reclamation of the facilities area.

Loose faces of slopes which are accessible will be "walked" with a dozer to partially compact the surfaces prior to coversoil placement. Haul and access roads to be reclaimed will be graded or have the edge berm pulled back prior to revegetation. In addition, compacted surfaces such as roads, parking areas and building areas, will be ripped or scarified (as discussed in Section 3.12.3.3) prior to coversoil placement and revegetation.

3.12.3.6 Revegetation

3.12.3.6.1 Coversoil Application. Prior to coversoil replacement, composite samples will be taken from each stockpile. The samples will be analyzed for texture, organic material, pH, and available macro and micro-nutrients. The analyses will be conducted at an approved state agricultural laboratory which will provide a fertilizer recommendation in addition to reporting analytical results. Coversoil will be placed on all reclaimed areas to provide growth medium for revegetation.

3.12.3.6.2 Species Selection. The selection of appropriate grass and forb species for revegetation, and appropriate sources of plant materials, is a critical process which controls the function of the overall revegetation program. All species included in the proposed seed mixtures are adapted to the area. Species selection was based on proposed land use, climate, and soil conditions. Specific seed sources will be selected to ensure that the plants are adapted to the elevation, precipitation, temperature, and soil conditions present at the Lamefoot site.

The species mixture chosen for revegetation is designed to provide a stable environment that is capable of supporting pre-mining land uses of wildlife habitat, recreation, and mineral exploration. All species selected for revegetation at the site are adapted to low fertility environments. Species that are adapted to low nutrient conditions are better suited to compete with aggressive weedy species during the critical plant establishment period. The utilization of adapted species and careful fertilization techniques will ultimately result in a plant community composed of species that can survive lower nutrient conditions, yet out-compete aggressive weedy invaders that require higher nutrient conditions. Manual spot applications of appropriate herbicides will be used to control limited outbreaks of noxious weeds in order to minimize loss of desirable forbs and shrubs (see Section 3.9). Seeds are available from suppliers in Washington State, and seedlings are available from suppliers in the Pacific Northwest who specialize in reclamation. Echo Bay will obtain seeds from established seed supply companies which produce seed adapted to the conditions at the Lamefoot site. Echo Bay will specify that plant materials originate from an elevation range of 2,500 to 3,500 feet and an average annual precipitation of 15 to 25 inches. If appropriate seed sources are not available, with adequate lead time, most seed distributors will collect appropriate seed sources as required for a given revegetation request.

The approved seed mixture proposed for use on private lands was developed by project soils and vegetation specialists in consultation with the Soil Conservation Service (SCS), BLM, and U.S. Forest Service (USFS) (Colville National Forest, Republic Ranger District) personnel for this and similar sites in the area. The seed mixture and broadcast application rate consists of:

•	2 lb PLS/acre	white Dutch clover (*Melilotus alba*)
•	4 lb PLS/acre	smooth brome (*Bromus inermes*)
•	4 lb PLS/acre	orchard grass (*Dactylis glomerata*)
•	4 lb PLS/acre	Sherman big bluegrass (*Poa ampla* var. Sherman)
•	4 lb PLS/acre	bluebunch wheatgrass (*Agropyron spicatum*)
•	4 lb PLS/acre	Idaho fescue (*Festuca idahoensis*)

The first four species are recommended because of their ability to colonize and quickly stabilize the soil on disturbed sites. The last two species are native to the Lamefoot area and will ultimately become the dominant species in the revegetated areas. However, these species are slow to colonize and, therefore, are not recommended for use alone in revegetation seed mixtures where erosion control is the overriding concern for the first year or two of growth.

The seed mixture and broadcast application rates for final reclamation of federal lands consists of:

• 3 lbs PLS/acre	mountain brome (*Bromus marginatus*)
• 3 lbs PLS/acre	slender wheatgrass (*Elymus trachycaulus*)
• 5 lbs PLS/acre	Idaho fescue (*Festuca idahoensis*)
• 5 lbs PLS/acre	bluebunch wheatgrass (*Agropyron spicata*)
• 2 lbs PLS/acre	canby bluegrass (*Poa canbyi*)
• 1 lb PLS/acre	serviceberry (*Amelanchier alnifolia*)
• 1/2 lb PLS/acre	Utah sweetvetch (*Hedysarum boreale*)
• 1/2 lb PLS	silky lupine (*Lupinus sericeus*)
• 1/2 lb PLS	arrowleaf balsamroot (*Balsamorhiza sagittata*)

The first two short-lived species are recommended for their seedling vigor and rapid establishment qualities. The next three species will likely become the dominant species as they are long-lived and are native to the Lamefoot area. The last four are included for diversity.

3.12.3.6.3 **Seeding and Planting**. If possible, all seeding and planting activities will be conducted in the fall at the conclusion of regrading, placement of coversoil, fertilization, and seedbed preparation. Seeding is most effective when completed prior to the period of peak precipitation. If fall seeding is unsuccessful, areas will be re-seeded in the spring. Seedbed preparation will be accomplished using an agricultural disk on the contour, when feasible. The surface of the prepared seedbed will be left relatively rough to create microsites to facilitate burial of seed and establishment of seedlings. Seed will be broadcast with a cyclone-type broadcaster on accessible sites, and by means of hand broadcasting and hydroseeding on steep slopes, as access allows. The seedbed will be harrowed or dragged following seeding to ensure proper seed burial. Broadcast seeding techniques will be used on all disturbed areas to create a more natural-appearing plant community. Broadcast seeding disperses seed over the soil surface in a random pattern. This provides a more natural-appearing plant community than drill seeding, which plants seeds in straight lines at fixed intervals.

3.12.3.6.4 Mulch Application. Straw mulch will be applied, based on criteria established in Section ~~1.1~~ 3.12.2, to seeded areas immediately after seeding to facilitate plant-community development, and to protect the seeded areas from wind and water erosion until the plants have become established. Mulch provides a favorable plant growth environment by maintaining effective soil temperatures to enhance germination and by conserving moisture in the soil profile for plant use.

3.12.4 Reclamation of Drill Sites/Roads

Reclamation of the Lamefoot drill sites and access roads will be performed incrementally. Those areas likely to be needed for access to ongoing exploration and/or full-scale mining activities will not be reclaimed until those activities are completed. However, adequate drainage patterns, erosion control berms, cresting of road surfaces, or construction of water bars will be established on these unreclaimed areas so as to avoid formation of erosion rills or gullies. Barriers and signs will be placed on reclaimed roads to discourage use by off-road vehicles (ORVs) on both private and public lands.

3.12.4.1 Slash Clearing and Coversoil Stripping and Salvaging

In selecting the exact drill pad locations for possible future exploration activity, Echo Bay will conduct its activities as previously permitted. Echo Bay will choose as many open areas as possible in order to minimize the disturbance to mature trees.

In the past, any slash that was present was isolated, piled, and left in place for establishment of microenvironments for rodents and small mammals. Where practical, coversoil (growth medium) was stripped to a depth of 12 inches from areas of the roads and drill pads. A total of 3,865 cubic yards of material was salvaged and stored in small stockpiles adjacent to each pad site or as sidecast along roadways. In areas where coversoil is to remain in storage for a period exceeding six months, the stockpiles are seeded with the temporary seed mixture.

3.12.4.2 Road and Pad Reclamation

Recontouring of drill pads to approximately pre-disturbance contours will be done to the extent possible at all Lamefoot sites whether on federal or private land. Many of the sites in flatter terrain will not require recontouring but may require coversoil addition. In areas with moderate slopes, recontouring can be accomplished by salvaging sidecast deposited during road construction. In areas with very steep slopes, some additional material will necessarily be imported in order to recontour.

Recontouring procedures for drill roads with cuts will be determined by the steepness of slopes, height of cuts, and size of sidecast. On gentler slopes, where roads occur in unconsolidated material, a bulldozer will be used along contours to reshape cuts by pushing up sidecast material. In areas with moderate slopes, a track-mounted excavator with a 16-foot reach will be used along contours to pull up sidecast prior to reshaping with the CAT. In areas too steep for sidehill operations of the CAT, the excavator will place the material and compact it using the bucket. These techniques will be used on all disturbance in areas identified in the 1992 Lamefoot Plan of Operations (Echo Bay 1992a) as B,C,D,E and F. It is anticipated that areas A and G will require no recontouring. Figures 28, 28a, and 28b show before-and-after reclamation contours.

Revegetation will use the seed mixtures presented in Section 3.12.3.6.2. Seeding will take place in the fall immediately following coversoil placement; and the surfaces will be broadcast-seeded and mulched using the techniques described in Section 3.12.2 and Section 3.12.3.6.3.

Concurrent reclamation will be performed on the middle and upper exploration roads in the area identified in the 1992 Lamefoot Plan of Operations (Echo Bay 1992a) as H. These roads will be reclaimed, when they are no longer needed, by pushing the existing berm to the inside of the road with a D-6 or D-7 CAT excavator, which will then pull sidecast onto the road. The CAT will push the reclaimed sidecast toward the end of the road, flattening it, as required for subsequent steps. Next, fill will be brought in by dump trucks and the excavator will place the fill and reshape topography to the extent possible. Fill material will consist of glacial gravel from site construction or broken rock from underground or surface excavation. Prior to placement of coversoil, a base application of fertilizer will be placed on the surface of the replaced waste rock. This will be done in short increments no longer than the reach of the excavators' arm. When each segment is finished, coversoil will be brought in and placed in a similar manner. This procedure will be repeated until the entire road length on BLM land is complete. Coversoil will be placed to a depth of approximately eight inches. The surface will be revegetated using the seed mixture presented in Section 3.12.3.6.2.

To protect public safety, some portions of Area H will remain unreclaimed where rock slopes are too steep to revegetate. This determination will be at the discretion of the field personnel. If open to traffic during this reclamation, the existing Wolfe Camp Road will be intermittently closed by flaggers to avoid the hazard of rolling rocks dislodged by the reclamation effort.

The steep slope above the Wolfe Camp Road is presently being stabilized through the use of screens. Additional activities to stabilize the slopes above Wolfe Camp Road are discussed in Section 3.12.6.3.

3.12.4.2.1 **Staining**. Echo Bay will experiment with rock staining to determine if natural looking results can be accomplished with the rock type present in the project area. Should the results of the staining tests be judged successful, exposed rock faces in Area H and other appropriate visible rock faces will be stained within the first two seasons of operations to reduce the visual contrast between the fresh cut rock surfaces and the natural landscape.

The stain will be an environmentally safe solution that is sprayed on the rock cuts that, over a one-week period, develops into earth-tone colors that can be varied by using different spray patterns and concentrations.

3.12.5 Drill Hole Abandonment

Completed drill holes are and will be abandoned during reclamation using the methods specified by WAC 173-160 Minimum Standards For Construction and Maintenance Of Wells. Prior to capping, all holes will be filled with a drill hole abandonment fluid consisting of a premium-grade bentonite mixed to a +50 viscosity. The surface casing will be pulled or cut off, and the top 20 feet of the hole will be cemented. If artesian water is encountered in a hole, that hole will be cemented from the bottom (T.D.) to the top (collar).

Accessible drill holes which intersect underground workings during exploration and production will be cemented if they promote an inflow of water.

3.12.6 Reclamation of Disturbances Associated With Underground Exploration

3.12.6.1 Surface Facilities

Appurtenant surface facilities have been constructed on private land, in association with underground exploration. These facilities will be removed and the surface disturbance will be reclaimed at mine closure.

In general, the disturbance areas have been cleared and stripped of coversoil. The coversoil was stockpiled, and the locations have been selected to protect the coversoil resource by minimizing wind and water erosion and facilitating later reclamation activities. The coversoil stockpiles have been seeded to provide a stabilizing cover to minimize erosion. Upon decommissioning, stockpiled coversoil will be redistributed on the various sites prior to revegetation. The coversoil stockpile locations are shown as growth media berms on Figure 6.

The tops of the coversoil stockpiles which parallel State Highway 21 have been planted with hybrid poplars (*Populus spp.*) to provide stabilization, and a visual and noise buffer for traffic on State Highway 21 and residents along the west side of Curlew Lake. The stockpiles have been seeded with an approved seed mix to provide interim reclamation. The trees will be removed when final reclamation is completed.

Upon final closure of the mine and completion of exploration activities, surface facilities are planned to be removed. Equipment from the site will be removed for use at other Echo Bay mines or sold in place. Remaining structures will be demolished and salvageable materials will be sold. Foundations of removed buildings will be broken up and buried in-place, in accordance with WAC 173-304 Minimum Functional Standards For Solid Waste Handling. The area will be recontoured to establish natural drainage patterns. Prior to placing coversoil, the surface will be ripped using a dozer-mounted ripper bar to a depth of 12 to 18 inches to alleviate compaction and increase infiltration. This practice will reduce runoff and erosion, provide more available moisture for plant roots, and provide a roughened contact surface for applied coversoil. After ripping, coversoil will be placed at a depth of approximately 8 inches. Coversoil will be re-spread using a bulldozer. Echo Bay has estimated that 235,000 ft^3 of coversoil are stored in the stockpiles at the surface facilities. This volume is sufficient to place approximately 8 inches of coversoil on the area occupied by surface facilities and on the estimated 2.3 acres of drill access road that will require hauling of coversoil from stockpiles at the surface facilities. Table 12 summarizes the coversoil mass balance. It has been assumed that the former waste disposal areas will not require coversoil placement following removal of the waste rock, because these areas were only partially stripped of coversoil. Textural analysis of material from the area to the south of the southern waste area, which has been partially stripped but not covered with waste, indicates there is sufficient growth medium to support revegetation.

Following coversoil placement, the surface will be shallow-ripped along the contour to a depth of approximately 8 inches, using a chisel plow. This will create a roughened surface which provides protected micro-environments to aid in vegetation establishment. It is anticipated that a stable vegetation community can be established without fertilization. If vegetation monitoring indicates symptoms of nutrient deficiency preventing successful revegetation, Echo Bay personnel will determine fertilizer rates which will allow establishment of a low-maintenance plant community. Additional samples will be taken from all surfaces to be revegetated and coversoil stockpiles, and will be tested for available nutrients prior to coversoil placement. It is likely that nutrient conditions may differ slightly from those currently existing on the site. Care will be taken to minimize the use of nitrogen, which promotes the establishment of weedy species.

Seeding will be conducted in the fall whenever possible to take advantage of late fall and early spring moisture. Reclaimed areas will be reseeded by broadcast seeding, using the approved seed mix. Broadcast seeding provides for a more natural-appearing community than drill seeding which plants seeds in straight lines at fixed intervals and results in a linear-appearing plant community. The pre-disturbance vegetative community on this site was a bluebunch wheatgrass/Idaho fescue community. Woody species will not be planted in this area in order to allow establishment of a vegetative community similar to the pre-mining community.

3.12.6.2 Waste Rock Disposal Area

At the conclusion of mining, it is anticipated that most of the waste rock will be returned to the mine by backfilling the material in the underground workings. Any remaining waste rock areas on the surface will be contoured to a slope of approximately 2.5 H:1 V. The waste rock pile will be revegetated after placement of a coversoil cover.

3.12.6.3 Underground Workings

Underground workings will be reclaimed by backfilling selected areas with material from the waste rock piles and the road relocation excavation. In the case of temporary closure of the mine, a steel door will be placed at the entrance to the workings to prevent entrance. Upon ultimate abandonment of the site, the portal will be plugged, recontoured, and revegetated (Figure 29).

Plugging of the adit will be accomplished according to the guidelines specified in the BLM Reclamation Handbook (1992b). Plugging the workings will also provide several environmental advantages, and preventing flow from the workings will allow the water table to stabilize at the pre-mining level. This will inundate some of the workings, thus minimizing oxidation of sulfide-bearing material, preventing acid drainage, and providing additional resistance to potential subsidence.

Backfilling will be the primary means of preventing subsidence of the rock overlying the slopes. This will serve the dual purpose of preventing surface damage caused by subsidence and minimizing flow into the workings by preventing fractures caused by subsidence.

The cut face of the entrance to the portal will be partially backfilled. The upper slopes will be left as benches and highwalls. Concurrent reclamation of the upper slopes is in progress and will continue. The slopes have been hydroseeded using the approved seed mix. An erosion control blanket has been placed on parts of the upper face to stabilize the surface and to provide a suitable micro-environment for revegetation.

The ventilation raise at mine closure will be permanently capped. The cap will be designed to prevent subsidence at the surface even in the event of collapse of the raise walls. As shown on Figure 30, the cap will be constructed of reinforced concrete. Figures 31, 32, and 33 present conceptual designs for permanent closure of the other adits at the Lamefoot Mine.

3.12.6.4 Sediment Control Structures

The diversion ditches are being reclaimed concurrently by revegetating with the approved seed mix. Vegetation will reduce sediment by stabilizing the sideslopes of the ditches and by reducing the velocity of flows in the ditch.

3.12.6.5 Key Mill Site

The Key Mill and tailings impoundment will be reclaimed as previously described, and as permitted in the Kettle River Project Final Environmental Impact Statement (DOE 1988). This site is on private land and was permitted by the Washington DOE.

3.12.7 Temporary Closure Plan

Although the proposed project will operate year round, it is possible that conditions may be encountered during the life of the project that may require temporary shutdown of the operations. The primary concern during a temporary closure will be maintenance of the water control structures. Because of the substantial investment Echo Bay will have in the facility, it will be staffed during any shutdown.

The temporary closure plan is based on an economic condition shutdown of more than one year's duration. The following steps will be followed to ensure that the closure and subsequent startup activities will be accomplished in a timely and environmentally safe manner:

- In the event of a temporary shutdown planned for more than one year, the appropriate regulatory officials in the BLM and Washington DOE will be promptly notified. Reclamation activities will proceed if the temporary shutdown exceeds more than 6 years.

- Inventories of petroleum products, and explosive or other potentially hazardous supplies that will not be used during shutdown will be used at other sites, returned to suppliers, or disposed of in compliance with the appropriate regulations.

- Echo Bay personnel will regularly inspect on-site shutdown monitoring and maintenance activities, including:

    - Maintaining access roads to all project facilities and monitoring locations

    - Maintaining diversion ditches and other facilities in preparation for snowmelt

    - Security and fire patrol of the facilities

- Monitoring inspection reports will be completed and submitted to appropriate state and federal regulatory agencies during the temporary closure period, as required.

3.12.8 Post-Closure Monitoring

3.12.8.1 Subsidence

During mining, the stopewalls and surface will be monitored for signs of subsidence. The results of this monitoring will be used to determine the placement of backfill during final reclamation.

The surface will be visually monitored annually for a five-year period following mining to ~~in order to~~ determine whether there is evidence of surface subsidence associated with the underground workings. Backfilling of the underground workings should prevent subsidence; however, in the unlikely event that subsidence is observed, Echo Bay will consult with the appropriate agencies to determine mitigation measures.

3.12.8.2 Acid Generation

Details for monitoring for acid drainage are discussed in Section 3.14 and Section 4.3 of the FEIS (USFS 1992).

3.12.8.3 Revegetation Monitoring Plan

The reclaimed private and BLM land will be visually inspected by qualified Echo Bay personnel to verify that revegetation is adequate to provide the post-mining land use of wildlife habitat. Inspections will be performed annually for a minimum of three years during the peak growing season until a satisfactory vegetative community has been established on all reclaimed lands. Criteria for a satisfactory vegetative community will be the achievement of 75 percent of cover and an average species diversity of 35 to 40 percent for a reference community to be selected with concurrence of the BLM. Echo Bay will monitor both the private and BLM land, and will repair any areas where revegetation is not successful or where excessive soil erosion is evident.

3.12.9 Public Safety

Various activities prior to and during the mine life and closures will be and have been performed to protect public safety. Most of these measures have been discussed in the main body of the POO. These measures include:

- Temporary stabilization of talus slopes above the old Wolfe Camp Road alignment by removing large boulders and netting the remaining material with chain link.
- Relocation of the Wolfe Camp Road junction with State Highway 21 to avoid traffic conflicts between the public and mine equipment.
- Prevention of potential public safety hazards associated with talus slopes above the old Wolfe Camp Road alignment by relocating the road to the opposite side of the drainage.
- Closure of underground workings by backfilling and plugging the portal, the exhaust cross-cut portal, and the ventilation shaft by constructing blockages at the entrances, as well as covering by backfilling to return the surfaces to approximate pre-mining contours.
- Recontour middle exploration road and upper road as a ditch (Area H).

3.12.10 Noxious Weed Control Plan

Reclamation will take place as soon as possible after disturbance, using a seed mixture which will include species that establish quickly. This will help prevent the establishment of noxious weed species. Only certified noxious weed-free species will be used in reclamation. Minimizing the use of fertilization will also prevent the establishment of noxious weed species which prefer high nitrogen levels in soil.

A list of noxious weeds that may appear on the Lamefoot site is presented in Table 13. The noxious weed control plan will include the application of appropriate herbicides.

Herbicide technology for the control of weeds has developed rapidly in recent years, and is the most widely used means of removing noxious weeds from naturally-occurring plant communities. The use of herbicides has many advantages over other control methods (including biological, mechanical, and fire methods). These advantages include:

- Less expensive than most mechanical methods
- Can be used on steep or rocky slopes where mechanical methods are not possible
- Provides a selective means of killing certain weed species, such as root sprouters that cannot be efficiently controlled by other methods
- Maintains grass and litter cover, and does not expose soil to erosion
- Provides rapid control
- Is safer than fire

The plant species proposed for revegetating all facilities include a mixture of grasses and forbs. The herbicides which are proposed for controlling all noxious weeds that may invade revegetated areas are 2,4-D and Tordon. These herbicides, when properly applied, will not damage grasses, but will affect broadleaf plants such as forbs and shrubs. Annual weed inspections and treatments will keep establishment of these species to a minimum and provide adequate control.

The use of herbicides on federal land must be approved by the BLM. Echo Bay will consult with the BLM to determine the most appropriate herbicide.

3.12.11 Underground Reclamation

Reclamation of underground workings by backfilling, plugging, and capping, will minimize surface subsidence and the potential for acid drainage, and will allow the groundwater levels to return to an approximate pre-mining level. Bulkheads will be required to prevent groundwater that enters the Lamefoot Mine from bypassing through the mine workings, to impound the water entering the Lamefoot Mine, and to reestablish approximate pre-mining groundwater conditions upon ultimate abandonment of the site. Closures will also be required to prevent human access to near-surface mine openings when the site is abandoned.

The goal of bulkhead construction is to eliminate the potential point sources for mine drainage at the mine portals and, thereby, to return groundwater entering the mine to the groundwater flow regime. The resulting mine pool will be in contact with limestone which should buffer the impounded mine water and reduce the oxygen available for the oxidation of sulfide minerals exposed in stope walls and in some of the backfill. Appendix I presents the design calculations for the underground bulkheads. Four bulkheads will be required to impound water that enters the mine, moves through low-resistance flow paths, and exits the access adit portal during the operating period. One bulkhead will be required in the access adit (Figure 29), one in the backfill adit (Figure 33), and one in each of the two the old prospect adits (Figures 31 and 32) that provide exhaust ventilation near the top of the mountain. The access adit bulkhead must be constructed at a sufficient distance from the portal to eliminate any potential for either chimney collapse to the surface or hydrofracture of the adit walls. The backfill adit bulkhead must be constructed east of Wolfe Camp Road, where sufficient cover will be present to eliminate any potential for either chimney collapse to the surface or hydrofracture of the adit walls. An exhaust-ventilation adit bulkhead will only be required if its elevation is below that of the highest elevation in the mine stopes, and could result in impounded water being forced out of that portal.

Bulkhead design to prevent point-source release of water impounded in the mine pool would be based on the potential head, opening dimensions, hydraulic pressure gradient, concrete shear stress, rock shear stress, one-way beam bending, hydrofracture potential, and potential chimney collapse height. The recommended closure design method is the American Concrete Institute, Building Code Requirements for reinforced concrete (ACI 318-89 rev 1992) plus hydrofracture potential (Einarson & Abel 1990). All of the bulkheads are overdesigned for 1.56 to 2.15 times the maximum possible head. This method has been successfully used during construction of the two acid-mine drainage bulkheads at the Summitville Mine in Southern Colorado (under EPA controls), two

o.k

bulkheads at the Comet claims above Silverton, Colorado (under Colorado Division of Minerals and Geology control), 14 bulkheads west of Redding, California (under California Department of Water Quality control), two bulkheads at the Sunnyside Mine at Gladstone, Colorado (under Colorado Division of Minerals and Geology control), and elsewhere. Calculations for the design of bulkheads at the Lamefoot Mine are presented in Appendix I.

Adit and shaft closures are safety measures to prevent access to the mine workings after closure of the mine, and are designed to resist hydraulic pressure. Shaft closures will be required over the top of the ventilation shaft east of Wolfe Camp Road, and over the top of the cemented and uncemented waste backfill shafts in the quarry west of Wolfe Camp Road. An adit closure will be required at the portal of the access adit to prevent human access to the Lamefoot Mine between the portal and the bulkhead in the access adit. A closure is recommended for the prospect adit only in the case where no bulkhead is constructed at that location. If a closure is installed in the prospect adit, it should be placed well inside the portal to maintain the approximate conditions present before the Lamefoot Mine exploration program.

3.12.12 Backfill Quarry Reclamation (adapted from Golder Associates 1994)

It is Echo Bay's intent to reclaim the pit and haul roads following the completion of mining operations. The various aspects of the reclamation plan are addressed in the following sections.

3.12.12.1 Timber Harvest and Site Clearing

Prior to the commencement of clearing and mining activities, any timber within the quarry perimeter will be logged as described in Section 3.12.4.1.

3.12.12.2 Coversoil Stripping and Salvage

Once the timber harvesting activities have been completed, coversoil will be stripped from within the pit perimeter and stockpiled for later use in reclamation.

Where practical, stockpiles will not exceed 30 feet in height in order to decrease wind erosion and compaction. Stockpile side slopes will not exceed 2 H:1 V in order to reduce wind and water erosion, and to enhance the growth of protective vegetation cover. The stockpiles will be seeded with the temporary seed mixture for private land as detailed in Section 3.12.2.

3.12.12.3 Quarry

The objective of the reclamation efforts for the quarry will be to ensure long-term overall stability of the walls. At the completion of mining, the configuration of the quarry will consist of 25-foot-high benches separated by catch-berms of varying widths. The catch-berms are intended to retain any ravelling or sloughing debris. The overall wall slopes will vary from 37 to 45 degrees, and the maximum height of the slopes will be approximately 125 feet.

Catch-benches will be constructed to ensure the surface will have an inclination toward the wall of the pit so moisture can be retained on the catch-bench for the purpose of revegetation. Approximately 8 inches of coversoil will be placed on the catch-berms during mining to the extent practical, in order to enhance revegetation of the slopes following the completion of mining. The benches will be revegetated with grasses and shrubs as described in Section 3.12.3.6.2 for private land.

Public safety on the relocated county road will be achieved by preservation of the benches in the quarry and adequate set-back of the highwall to at least 50 feet. However, the layout of the location of the blast holes will be designed so the benches will have a sinuous outline in plan view. (The exact outline of the benches will be finalized through the surface mining permit process, and the ~~D~~iagrams of the quarry are conceptual. ~~they will be redrawn to incorporate a sinuous outline~~. The quarry configuration has been ~~in being re~~designed with the Division of Natural Resources and the County Engineer in accordance with their guidelines and will be approved by the County Commissioners.) Berms approximately 6 feet high with 1.5 H:1 V side slopes will be constructed, where possible, around the perimeter of the quarry about 30 feet back from the edge. In areas where site topography precludes the construction of rock berms, appropriate weather-resistant warning signs will be posted at regular intervals stating

"Warning Steep Slopes Ahead." Any roads that are intersected by the quarry cut will be blocked by earth berms to prevent occasional ORVs from driving over the highwall.

3.12.12.4 County Road Relocation

A primitive county road that is used by local residents is located along the east side of the valley. The development of exploration roads on the slope above the road has resulted in slope instability. Following the completion of mining operations, Echo Bay proposes to relocate the county road along the in-pit haul road alignment that will be established along the east edge of the pit floor. The location of the haul road alignment is shown in Figure 24 and detailed plan and profile views of the relocated county road are shown in Figures 24a, 24b, 24c, and 24d.

The old county road may be used as a pad for a post-closure monitoring, in which case a gate will be used to permanently close off the road. Otherwise, access to the old county road will be cut off by placing berms across the intersections of the old road and the new road.

3.12.12.5 Closure Criteria

Echo Bay will cover the benches with top soil and revegatate them concurrently with operations. Immediately following the end of operations, Echo Bay will transfer the operating permit to Ferry County.

3.13 WATER MANAGEMENT AND MONITORING

Echo Bay has incorporated all known and reasonable technologies into the design of mitigation measures that will minimize both water inflow to the mine and degradation of water quality of any water that may remain in the workings. The following sections detail several water control and management options.

3.13.1 Grouting

In order to minimize the potential for discharge of mine water into the outside environment, all underground sources of water inflow will be grouted to halt or reduce that inflow. Water may enter the mine workings through naturally occurring discontinuities in the rock mass, including faults, joints, and lithologic contacts. Water may also enter the mine through drill holes which connect to the surface or intersect natural discontinuities. It is the intention of Echo Bay to fill these drill holes and discontinuities with grout.

Naturally occurring discontinuities exposed in the accessible mine workings, which are producing in excess of 1 gpm, will be grouted to the extent possible until the flow stops or becomes less than 1 gpm. This will be accomplished by drilling holes around the point of inflow with a percussion drill to intersect the natural discontinuity, then by pumping grout through the drill holes into the discontinuity. Inspections for sites of inflow will be conducted on a bimonthly basis, and written records of these inspections will be kept at the mine site.

All diamond drill and rotary drill holes will be filled, if possible, with grout to stop existing inflow and to prevent any future inflow. Holes at an angle above the horizontal plane will have a packer inserted at the collar, and grout will be pumped through the packer to fill the hole. A breather tube will be inserted through the packer so trapped air can escape from the hole, allowing the hole to be fully grouted.

In holes drilled horizontally or downward-dipping, a tube will be inserted to the bottom of the hole. Grout will be pumped down the tube. Pumping grout down the tube to the bottom of the hole will allow any water and air in the hole to be forced out the collar, ensuring a fully grouted hole. When grout begins to come out of the collar, grouting will be stopped and the tube will be cut off at the collar.

In the event that, after grouting the remaining inflow of water into the mine is not sufficient to meet the water requirements of the mining operation, a few holes will initially be sealed with mechanical packers and remain ungrouted. These holes can then be used as a source of additional mine water to meet mining requirements. At the end of the mining operation, these holes will be grouted through the packer.

Grouting equipment includes a Cominco Inc. Model MD 122 grout pump. Grout will be Type 1 cement and water, mixed to a very thick consistency.

A program to grout or seal all drill holes and discontinuities producing greater than 1 gpm, encountered in workings during underground exploration, will begin prior to the mine going into commercial production. Once commercial production begins, drill holes and discontinuities which produce water will be routinely grouted.

3.13.2 Surface Water Controls

3.13.2.1 Diversion Above Mine Workings

Purpose

A roadside interceptor ditch will be constructed to divert surface water runoff from hill slopes above the mine workings to the valley bottom of the Wolfe Camp Road drainage. This diversion will collect hill slope runoff that would normally flow downslope onto the land surface directly above the mine workings. By capturing this storm runoff, surface water seepage into soils above the mine workings will be minimized, and subsequent migration of groundwater toward the mine will be minimized.

Design

Designs for the interceptor ditch are divided into three reaches to account for significant changes in hydrologic and hydraulic conditions. Figure 34 shows the location of the structure and the extent of each of the three reaches, along with the location of a culvert that will carry flow under Wolfe Camp Road. Runoff estimates used in the design of the ditch were performed using the SCS Curve Number Method.

Reach 1 takes advantage of an existing road that was built during pre-mining exploration. The existing road cut will be reconfigured so that the road cross section gently slopes toward the hillside at a 1:15 gradient, thus forming a skewed V-shaped channel. The hillside immediately adjacent to the channel will be configured to a 1:2 gradient. Figure 35 shows a typical cross section for Reach 1 and the approximate flow level resulting from the 100-year, 24-hour storm event. Flow parameters were calculated with the aid of an open-channel flow computing program (Flowmaster).

Design specifications for Reach 2 are shown in Figure 36. This design uses rock riprap to provide long-term erosion protection that will be necessary in this reach due to the steepness of the channel slope. It is anticipated that rock material of adequate hardness will be readily available from the mine workings. Screening equipment will be used to properly size the riprap material. A filter fabric will be installed on surfaces that will be covered with riprap to retard erosion of base materials as water flows between the larger rock clasts during major flow events.

Reach 3 will require a design nearly identical to that presented for Reach 1. The Proposed design for Reach 3 is shown in Figure 37. At the end of Reach 3, storm runoff will flow into an existing roadside ditch on the east side of Wolfe Camp Road. Flow will continue in this ditch until it is conveyed under the road by a culvert that will have a minimum capacity to convey the 25-year, 24-hour flow event. The approximate culvert location is shown in Figure 34. A corrugated, smooth-walled interior, high density polyethylene pipe (HDPE) will be used along with fabricated polyethylene flared sections for both upstream and downstream ends. Figure 38 shows the conceptual design along with a culvert outlet riprap apron for erosion control.

Erosion Control

To ensure that this diversion remains functional during operations, each channel section will be inspected after major storm runoff events. Indications of erosion or downcutting will be repaired with gravel, riprap, or other materials as each case may require. In those rare events where channel erosion in Reaches 1 and 3 does occur, one of three measures shown in Figure 39 may be instituted.

- Gravel or rock bottom channel - Gravel or crushed limestone from the mine workings will be spread uniformly over the channel flow area to retard soil erosion in those local and particularly erodible areas.

- Rock berms - Check dams made of limestone or non-reactive rock, not exceeding two feet in height and spaced approximately 20 feet apart (based on 11 percent channel slope) in channel Reaches 1 and 3, will provide additional protection when and if needed.

- French drain - A 10-inch-diameter slotted pipe installed in a gravel-filled trench in the channel bottom will provide an alternative whereby most channel flow generated by the 25-year storm event would be conveyed within the pipe after surface runoff infiltrates into the gravel and flows into the pipe. Figure 39 shows a generalized cross-section design for this measure in Reach 1 only.

3.13.2.2 Backfill Quarry Retention Berm

Purpose

The location map in Figure 24 shows the extent of the quarry retention berm. The berm will be constructed adjacent to the quarry area to isolate storm runoff from the disturbed hill slopes, thus minimizing any impacts to downstream water quality. The upper end of this berm will divert flow in the Wolf Camp Road valley bottom and away from the quarry area. [illegible] While the quarry is active surface water runoff will be controled by a road side ditch and berm system similar to that shown in Figure 24, ~~although~~ but the berm may be temporarily located on the east side of Wolf Camp Road to allow unimpeded access to the quarry ~~with~~ while it is active.

Design

The structure will consist of berm with 3:1 side slopes extending the length of the quarry and cement plant area. It will begin upstream of the disturbance area in the valley bottom above the cement and rockfill facilities, and will extend down the eastern edge to a point on the valley side below the area of quarry disturbance. Valley cross sections shown in Figures 25 and 26 show the approximate relationship between the retention structure and the quarry area. Design details for the quarry retention berm are shown in Figure 40. The berm will be constructed to a height of 2.5 feet. A valley bottom flow conveyance area will be defined on the up-valley end of the berm on one side and the valley wall on the other. The minimum width of the conveyance will be 12 feet in this area. As Figure 40 shows, the 100-year, 24-hour flow event will pass without overtopping the berm and entering the quarry area.

Runoff estimates used in this design were performed using the SCS Curve Number Method. Total runoff volume from the quarry area was calculated to ensure that the berm will impound all runoff that will result from the 100-year, 24-hour precipitation event. Flows from the Wolfe Camp Road drainage above the quarry for the 100-year and 25-year storm events were also estimated using a curve number-based method (HEC I). Although ponding will probably seldom occur behind the berm, and flow will rarely be observed in the conveyance area, these structures will provide ample protection when and if unusually large storms develop over the area during the active mining period.

3.13.3 Underground Water Use

Echo Bay has completed a preliminary study to address any adverse impacts from the quality of water generated from the underground workings. This study evaluates water treatment options, with a special emphasis on nitrates in water.

Although water treatment is an option, there are several source control or waste minimization measures that Echo Bay will be implementing within the mine to reduce the levels of nitrate and other analytes in discharges to the infiltration pond. These pollution prevention measures include:

- Use of alternate explosives that contain lower amounts of nitrates
- Minimization of contamination of waters
- Segregation of waters

These source control or pollution prevention alternatives would be implemented in three phases. Phase I will involve control of the water source. The measures undertaken in this phase include:

- Grouting cracks and seeps
- Controlling infiltration and acid-forming conditions
- Reusing mine water at the cement batch plant
- Reusing mine water as make-up water for drilling
- Reusing water for dust suppression within the mine if it meets DOE standards

Phase I and Phase II (discussed below) will be conducted concurrently with the startup of operations.

3.13.4 Alternate Blasting Compounds

Echo Bay is currently experimenting with the use of low-solubility blasting compounds. Phase II would involve changing the blasting compound to an emulsion product. A report has been provided to BLM and DOE detailing the alternate compounds.

Several other mines in the U.S. and Canada have utilized emulsions to address nitrate issues.

3.13.5 Water Treatment

Should there be any water remaining that contains nitrates or other contaminants exceeding permit limits, the following options will be considered in Phase III:

- Physical, chemical or biological water treatment
- Artificial wetlands
- Land application
- Alternate disposal of mine waters

The remainder of this section discusses the options mentioned above, and the feasibility and relative costs of each. The discussions are taken from studies commissioned by Echo Bay through Golder Associates (Mudder 1994, Unifield Engineering 1994).

For the purpose of nitrate removal technology evaluation, the following design parameters were used:

- Flow rate was assumed to range from 75 to 100 gpm
- Nitrate concentrations were assumed to be an average of 25 ppm and a maximum of 100 ppm
- Sulfate concentrations were assumed to be a maximum of 500 ppm
- Ambient temperatures were assumed to vary between -40°F and 100°F
- Water temperatures were assumed to vary between 33°F and 70°F

Several types of water treatment methods have been used or are being developed for removal of nitrates from water. These include:

- Ion exchange
- Biological denitrification
- Electrochemical ion exchange
- Combinations of these technologies

Ion exchange and biological denitrification are conventional removal technologies, whereas the electrochemical ion exchange is an emerging technology. In addition, artificial wetlands and land application are potential alternatives to treatment, and are variations of biological denitrification.

Ion Exchange

Ion exchange is the process by which soluble ions (either negatively or positively charged) are removed from solution by replacement with ions of a different species. Ion exchange resins can be of either strong or weak-base anionic (negatively charged ions) or strong or weak-acid cationic (positively charged ions) form. Anionic resins are used to exchange anions in solution, while cationic resins are used to exchange cations in solution. Strong acid or base resins work effectively over the entire pH range, while weak acid resins work effectively only above a pH of 7 s.u. A weak base resin is effective at a pH below 6 s.u. (Unified Engineering 1994).

A typical block flow diagram for an ion exchange process in shown in Figure 41. Depending on the concentration of total suspended solids (TSS), pretreatment ahead of the ion exchange unit may be necessary. This pretreatment would probably consist of clarification and/or filtration, depending on TSS concentrations. The ion exchange unit is typically a packed bed in which feed water and regenerative brine flow from top to bottom. The process of ion exchange consists of feeding water through the system until the resin has been exhausted, then regenerating the media with a brine until the exchanged ions from the water have been removed. The length of time required between resin regenerations is dependent on the amount of resin present in the exchange column or bed, and on the concentration and flow of ions entering the system. The treated water can be discharged directly while the regenerative brine is typically further treated.

The use of ion exchange for removal of nitrate, as well as other cations and anions, is well established on a commercial basis. In practice, the synthetically prepared ion exchange resins are either placed in a fixed bed or suspended in a reactor as a slurry. The placement of the ion exchange resins in a fixed bed is the more common configuration, and is similar in design to a filtration system.

The advantages of this technology include:

- Relative ease of automation
- Excellent correlation between bench or pilot scale results with full scale performance
- Reliability with respect to performance and fluctuating flow and nitrate levels
- Relatively short time frame (as compared with other technologies) in which a full-scale system could be designed and placed into operation

The disadvantages include its relatively high capital and operational costs, and the need to backwash the resins, which produces a concentrated brine. The brine must then be disposed of in an environmentally acceptable manner. Due to the very high solubility of nitrate, on-site disposal is an ongoing problem because of the mobility of the anion and the possibility that it may reappear in other surface water or groundwater. Off-site disposal solves the problem of re-mobilization of the nitrate brine, but at a much higher cost, stemming from storage, transportation, and the acquisition of an additional disposal site.

One further disadvantage is the potential need to both clarify and filter the mine water prior to ion exchange treatment for nitrate removal. In the event that the TSS level of the mine water routinely exceeds 10 mg/l, incorporation of pressure filtration into the process design could become necessary, adding incrementally to the overall capital and operating costs for ion exchange treatment, due to the additional equipment needed and the added disposal requirements. TSS removal has been evaluated as part of the pretreatment system for removal of sulfates (SO_4), TSS, total dissolved solids (TDS), and metals discussed later in this section.

The capital and operating costs are influenced to a large extent by the price of the resin itself and the need to annually replace a portion of resin. The extent of resin replacement is controlled by the unavoidable losses that occur during backwash, and due to the physical destruction and permanent fouling of resin. Typically, the annual resin replacement will range from 10 to 20 percent of the entire bed. A complete ion exchange system could include an ion exchange unit; an approximately 20-foot-diameter pretreatment clarifier to remove suspended solids; a 2,000-cubic-foot storage silo for salt; and miscellaneous expenses associated with a building to house the ion exchange unit and controls to operate the complete system.

Capital cost for a complete ion exchange system could be approximately $400,000, excluding pretreatment costs. Operating cost for the system could be approximately $0.15 per 1,000 gallons of water treated, excluding brine disposal and storage costs.

Biological Denitrification

One of the most common processes used for removal of nitrates from municipal wastewater treatment plant effluent has been biological denitrification. This process consists of an anoxic (oxygen-free) stage, with addition of a carbon source such as methanol, to convert the nitrates to gaseous nitrogen. Nitrate conversion is accomplished by bacteria present in the anoxic stage. For efficient conversion of nitrate at low organic carbon concentrations, methanol must be supplied to the process at a stoichiometric ratio of between 2.5 and 3.0 mg/L per mg/L of nitrate. The process is truly destructive, and there are no secondary disposal issues other than a biological sludge. The process is well-established and forms an integral component of many municipal wastewater treatment plants which require stringent control over discharges of nutrients.

Since the process involves living organisms, it is more sensitive to toxicants and fluctuations in mine water chemistry than is a physical or chemical treatment technology. The mine water chemistry may require adjustment through pretreatment to create an environment suitable for biological denitrification.

There are several disadvantages associated with biological denitrification. The first relates to the need for a supplemental organic carbon source, a requirement best fulfilled by addition of methanol. The need to supply methanol in pure form has a large impact on increasing the overall operating costs for the process.

Another disadvantage centers on the sensitivity of the process to upsets, which could occur due to the constituents contained in the mine water or frequent changes in its chemistry or temperature. The degree of nitrate removal is highly variable depending on water chemistry factors. Due to the variety of organic compounds (i.e., grouting materials or drilling fluids) and metals contained in the mine water, process upsets could occur, resulting in prolonged periods of poor performance. In addition, denitrifying bacteria may be inhibited or destroyed by exposure to substances such as heavy metals present in the mine water.

Two process designs are envisioned for biological denitrification and treatment of mine water. The first involves a conventional, attached-growth process system, utilizing either a submerged rotating biological contractor (RBC) or a biological filter. Both of these systems involve conventional off-the-shelf equipment and technology available through a variety of vendors and manufacturers. The second design would be an in-situ type system.

An RBC consists of large diameter, corrugated plastic media mounted on a horizontal shaft and placed in a concrete tank. A block flow diagram of the system is shown on Figure 42.

The RBC can either be partially submerged in water to achieve aerobic conditions or fully submerged (SRBC) to promote anaerobic conditions. After startup of the system, bacteria either normally present in the wastewater or artificially seeded adhere to the rotating surfaces and multiply. After about a week, the surface is covered with a thin film of biomass. As the biomass grows, it continually sloughs off into the wastewater. The biomass will raise the suspended solids concentration in the treated water effluent, which may then require clarification, depending on effluent suspended solids limitations.

Methanol requirements for the process can be a significant portion of the overall operating cost. The methanol which provides the carbon source for nitrate conversion, must be carefully regulated to ensure an excess of methanol does not enter the system and contaminate the effluent with excess carbon which could cause biological oxygen demand (BOD) effluent limitations to be exceeded.

A complete rotating biological contractor system could include an RBC unit; an approximately 20-foot-diameter post-treatment clarifier to remove excess suspended solids; and miscellaneous expenses associated with a building to house the unit and controls to operate the complete system.

Capital cost for the system could be approximately $500,000. Estimated operating cost for the SRBC system, including the cost of methanol needed for the unit, manpower, and other miscellaneous minor costs, could be $0.21 per 1,000 gallons of water treated.

Another type of biological filter is the high rate fixed-film filter. A block flow diagram of the system is shown on Figure 43. These systems typically employ a 6-foot-deep bed of silica sand media contained in a concrete or steel tank. Denitrifying bacteria grow on the media and in the media void spaces. Specially sized and shaped media are selected to ensure a high surface area for bacterial growth, maximum solids-holding capacity, and effective backwashing capability with minimal media loss or deterioration.

The system is normally operated in a downflow mode at a rate of 1 to 4 gpm/ft^2 when inflow nitrate concentrations are less than several hundred parts per million. When higher concentrations of nitrates are present, an upflow configuration is used to promote self purging of byproduct nitrogen gas. During normal downflow operation, nitrogen gas will accumulate in the media, causing an increase in system head loss. The excess gas is removed by taking the filter out of serve and applying a backwash rate of 8 gpm/ft^2 for several minutes. This process is referred to as "bumping." Bumping releases the entrapped nitrogen gas to the atmosphere and reduces system head loss. The frequency of bumping depends on the rate of denitrification occurring in the filter.

Since the system has a high capacity for retaining solids in the media, it requires a heavy-duty backwash to remove accumulated solids. Air and water are used simultaneously to create a vigorous backwashing action while minimizing backwash water volumes. A final backwash consisting of water only is used to remove entrained air before placing the filter back in service.

A complete high rate fixed-film biological filter using and intermediate backwash system could include a biological filter unit; an approximately 20-foot-diameter pretreatment clarifier to remove suspended solids; a post-treatment filter to remove excess suspended solids; and miscellaneous expenses associated with a building to house the unit and controls to operate the complete system.

Total estimated capital cost for the complete system could be approximately $770,000. Estimated operating cost for the system, including the cost of methanol needed for the unit, manpower, and other miscellaneous minor costs, could be $0.21 per 1,000 gallons of water treated.

The second process design is not conventional and would involve development of an in-situ type system, in which the current disposal method would be used with the addition of methanol and nutrients, as needed. In this case, the soil column itself would serve as the biological reactor, and the nitrate would be removed as the mine water passed downward through the ground. Such an approach has appeal due to its simplicity both in design and cost. However, this approach is novel and unique, and would require extensive laboratory and field testing to demonstrate its viability. There are also concerns relating to the examination of groundwater contamination and permitting of the process.

Electrochemical Ion Exchange

The electrochemical ion exchange process is an emerging technology, which has not been installed in large-scale applications. Limited information on the technology exists, and apparently only one vendor has developed and supplies the equipment. The process shown in Figure 44 combines nitrate removal by ion exchange with electrochemical treatment in order to move nitrate ions to an electrolytic compartment where they are converted to nitrogen, oxygen, and water. The primary disadvantage of this process relates to the relatively high operating costs due to the power consumption.

The advantages of the electrochemical process are that there is no generation of brine, and the unit is modular and mobile. Disadvantages which are similar to ion exchange include potential resin fouling, more complex system to operate due to multiple unit operations, potential pretreatment requirements, and production of gases that may require control.

The complete electrochemical ion exchange system could include an electrochemical ion exchange unit; an approximately 20-foot-diameter pretreatment clarifier to remove suspended solids; and miscellaneous expenses associated with a building to house the unit and controls to operate the complete system.

Total estimated capital cost for the complete system could be $590,000, excluding pretreatment costs. Estimated operating cost for the electrochemical ion exchange system, including the electrical cost to operate the unit, manpower, and other miscellaneous minor costs, could be $0.15 per 1,000 gallons of water treated.

Consideration of the technology centers on a possible unique application, which would involve its combination with ion exchange. The positive aspects of both technologies are used as a result of their combination, and a possibility exists for reducing the overall capital and operating costs for nitrate removal. The application would involve concentrating the nitrate as a brine, following removal from the mine water, using ion exchange. The brine would then be subject to electrochemical destruction as a concentrated waste. Using this approach would reduce the size and capacity of the equipment needed, and the need for disposal of a brine would be eliminated.

This particular technology would require the greatest amount of research and, at a minimum, a short pilot plant or continuous treatment evaluation. Information regarding the availability of pilot plant equipment may be sought during a later study phase. As with ion exchange, there may be a requirement to filter the mine water prior to electrochemical destruction.

Artificial Wetlands

Artificially constructed wetlands simulate natural areas such as marshes, bogs, wet meadows, peat lands, and swamps. Such an area could be constructed specifically for use in water treatment. The construction involves an impervious membrane under the soil substrate. Grasses or other wetlands vegetation are then planted and allowed to proliferate. If the wetland is constructed on a gentle slope, waters with undesirable quality are applied to the top of the wetland and allowed to flow slowly to the bottom where they are collected for final disposal with improved quality. The wetland will likely produce intermittent outflows, which would be allowed to infiltrate at the discharge point or would be routed elsewhere. Such wetlands have been effective in removing BOD and TSS, but most systems have been relatively ineffective in removing ammonia nitrogen. The primary limiting factor is believed to be dissolved oxygen (Watson and Danzig 1993).

Nitrate transformation is closely tied to biological activity and other factors (Montgomery 1985; also see the Biological Denitrification discussion above). Elevated nitrate levels are not rapidly and effectively attenuated (reduced or weakened) by flow through soil, rock, and groundwater (Watson and Danzig 1993). Further, a passive wetland treatment system may eventually become clogged and coated with sediments, thereby reducing its effectiveness if not periodically cleansed and rehabilitated (Holm 1989). Therefore, it is not clear whether such a system would effectively address nitrates and other constituents on its own. However, construction, operation, and maintenance costs would likely be low compared to the more active water treatment methods described here.

Land Application

Land application, which includes infiltration and irrigation, allows several processes to occur that can lessen the impact of nitrogen on water. Infiltration of nitrogen-rich water to groundwater provides some dilution and tends to attenuate or smooth out peaks in nitrogen concentration. Under aerobic conditions, nitrification will occur, oxidizing more toxic ammonia and nitrite to less toxic nitrate. Under anoxic conditions, denitrification can occur, releasing nitrogen gas to the atmosphere, and reducing the nitrogen content of the water. A soluble organic carbon energy source is necessary for denitrification to occur. Ammonium may be removed by adsorption on the cation-exchange complex of the soil.

Irrigation will result in the uptake and storage of some nitrogen by plants. Maximum plant uptake is about 100 to 700 kg N ha^{-1} y, depending on the crop, and the crop must be removed to achieve permanent N removal. In addition, the climate, soils and topography of the area must be suitable for the application of the quantity of water necessary to achieve the desired N application (Pommen 1983).

In general, land application would involve direct infiltration into soils, gravels, and bedrock to attenuate nitrates and other contaminants, using the cleansing capabilities of the environment. The soils, glacial gravels, and other substrates between the mine facility and Curlew Lake could be used for this purpose. Sprinkler/spray assemblies could be constructed to spray contaminated wastewater over suitable substrate. Once again, operation and maintenance costs are low compared to active treatment. Non-degradation of surface water and groundwater is a policy issue with the EPA and some states, however (MDSL et al. 1992).

Alternate Disposal of Mine Water

Alternate nitrate discharge mitigation methods may include transportation of mine effluent to off-site disposal (e.g., in the Key Mill tailings pond), subject to approval of the Department of Ecology.

Pretreatment of Mine Water

As mentioned in the description of nitrate removal technologies, pretreatment of mine water may be required to remove SO_4, TDS, and metals (As, Fe, Mn, etc.) prior to nitrate removal. The purpose of the pretreatment would be to reduce the above constituents to acceptable levels prior to discharge, and to aid in a more effective removal of nitrates. Each of the various constituents in the water can be removed effectively, but this requires a combination of processes. One potential pretreatment system involves several steps which are shown on Figure 45. These are:

- Chemical Addition
- Precipitation
- Clarification
- Green Sand Filtration
- Sludge Thickening
- Dewatering (filter press)
- Reverse Osmosis

Chemical addition would involve feeding chemicals such as lime or caustic to increase pH and precipitate particular metals. Oxidation chemicals such as peroxide or potassium permanganate would be added to oxidize and precipitate Fe, Mn, and to coprecipitate As. Polymers would be added to enhance suspended solids removal. Precipitation would be followed by filtration, and the solids would be collected in a tank and allowed to thicken. The sludge would then be dewatered, producing a cake for on-site or off-site disposal. After filtration, the water would pass through a reverse osmosis unit for further removal of suspended and dissolved solids. Approximately 10 percent of the flow becomes a concentrated reject stream. This stream would be combined with the sludge from the clarifier, which would be allowed to thicken and be dewatered, producing a solid for on-site or off-site disposal.

The pretreatment system could be added to the front end of each of the nitrate removal alternatives (ion exchange, biological denitrification, electrochemical). Although the potential removal of some nitrates in the pretreatment step would reduce the size, configuration, or cost of the nitrate removal alternatives, a conservative assumption for this evaluation is that nitrates will not be removed in the pretreatment step.

The advantage of this system is its flexibility in adjusting treatment if water quality changes. Constituents are also removed prior to the nitrate removal, thus enhancing the effectiveness of that process.

The disadvantages are that the pretreatment system involves multiple-unit operations and is equipment-intensive, requiring tanks, pumps, mixers, materials-handling equipment, instrumentation and controls, and chemicals. Operation and maintenance of the system would be labor-intensive and complex due to the numerous unit operations.

The capital cost of a pretreatment system has been estimated to be approximately $2 million, with $1.2 million accounting for the major pieces of equipment, installation, piping, building, etc. Operations cost for a treatment system such as this has been estimated to be approximately $290,000 per year, assuming that the sludge byproduct could be disposed as a non-hazardous waste.

Water quality discharge requirements shall be established by the Washington DOE (during the waste discharge permit review) and BLM (during the ROD phase of the FEIS).

3.14 HYDROLOGIC MONITORING PLAN SUMMARY

This section of the POO summarizes major points of the Hydrologic Monitoring Plan (HMP); the full text and supporting documentation for the HMP are attached as Appendix J.

Washington Department of Ecology (DOE) Waste Water Discharge Permit No. 8033 addresses monitoring at the Lamefoot Mine infiltration pond, Key Mill and Tailings Facility, and three other Key Project mines. DOE Permit No. 8033 is attached as Appendix D. The HMP for the Lamefoot Mine incorporates the relevant monitoring and reporting requirements of DOE Waste Water Discharge Permit No. 8033 (i.e., Sections S2, S3, S4, and S7), as well as additional monitoring that has been required or recommended as mitigation measures by the BLM. These requirements were established through the processes of permitting the Lamefoot exploration activities and preparation of the DEISS and the FEISS for the Lamefoot Project (BLM 1994).

The HMP addresses the monitoring and reporting activities throughout the operational period and after closure of the Lamefoot Mine. In particular, the HMP addresses the monitoring locations and sampling frequencies for mine water discharge, surface water (i.e., springs and wetlands), and groundwater at the mine site, as well as recordkeeping and reporting requirements. The HMP includes the following items:

- Map of hydrologic monitoring locations
- Construction specifications for proposed wells
- Aquifer-testing procedures
- Monitoring frequency
- Sample collection procedures
- Chemical parameters and analytical methods

Standard operating procedures (SOPs) for equipment decontamination, drilling and well installation, aquifer testing, sampling, and documentation are attached as part of the HMP. The SOPs were established to guide field personnel during the implementation of the monitoring program and assure the collection of consistent, reproducible, and defendable data.

Table J-1 (Appendix J) presents the frequency for monitoring discharge, water levels, and water quality under the HMP. Tables J-2 and J-3 present the analytes and methods of laboratory analyses for the water samples to be collected. The complete list of parameters will be analyzed for the laboratory sample locations and frequencies shown on Table J-1 for the first year of monitoring. During subsequent years of monitoring, the complete list will be analyzed once per year, and a short list of parameters (as indicated in bold on Tables J-2 and J-3) will be analyzed for intervening samples. Table J-4 outlines a hypothetical post-closure hydrologic monitoring plan.

3.15 OPERATIONAL SAMPLING OF BACKFILL AND WASTE ROCK

3.15.1 Backfill Quarry

Backfill quarry materials will be sampled at a frequency one sample per 10,000 tons. Given that the quarry materials to be backfilled total approximately 1,350,000 tons, 136 samples will be sampled for acid-production characteristics.

The first 50 samples will be analyzed for AGP, ANP, NNP, and Total Sulfur. Such results would allow estimation of the relationship between NNP and Total Sulfur. Once estimated, the remaining samples would be analyzed for Total Sulfur (%) only.

3.15.2 Underground Waste Rock

Underground waste rock will also be sampled at a frequency of one sample per 2,000 tons of waste rock. Since the percentage of waste rock will vary by area, the density of the holes sampled will also need to be adjusted by area. This would result in about 125 samples analyzed for Total Sulfur only (assuming approximately 250,000 tons of underground waste) over the 7-year life of the mine, at about 40 samples per year.

3.16 RECORDKEEPING AND REPORTING

Recordkeeping and Reporting. Recordkeeping and reporting will be in accordance with the requirements of DOE Permit No. 8033, and will document procedures presented in the HMP. Appendix J summarizes the detailed water quality monitoring planned. All records of monitoring activities and results shall be retained for the operational life of the mine, should any further actions become necessary. Records and results will be kept in the files at the mine site.

Operational monitoring results will be reported to the Washington DOE and the BLM on a monthly basis. Reporting and filing will be in accordance with the requirements of DOE Permit No. 8033 (Sections S3 and S7). The reports will present the tabulated measurements and the laboratory analytical results. In the case of post-closure monitoring, reports will tally and summarize results from previous efforts; for example, quarterly monitoring and reporting would be performed during the first 10 years after mine closure, based upon the schedule presented in Appendix J. Thereafter, annual or periodic interpretive monitoring reports will also present a discussion of monitoring data, water quality and quantity trends, and effectiveness of mitigation. Report information will include, but will not be restricted to, text, tables and/or maps describing:

- All environmental monitoring data
- Summary of mine development activities
- Summary of surface construction activities
- Summary of reclamation activities
- Summary of operation schedule
- Production records

Figures 46 and 47 show the monitoring plan, and the project and monitoring evaluation schedule.

indicated in bold on Tables J-2 and J-3) will be analyzed for intervening samples. Table J-4 outlines a hypothetical post-closure hydrologic monitoring plan.

3.15 OPERATIONAL SAMPLING OF BACKFILL AND WASTE ROCK

3.15.1 Backfill Quarry

Backfill quarry materials will be sampled at a frequency one sample per 10,000 tons. Given that the quarry materials to be backfilled total approximately 1,350,000 tons, 136 samples will be sampled for acid production characteristics.

The first 50 samples will be analyzed for AGP, ANP, NNP, and Total Sulfur. Such results would allow estimation of the relationship between NNP and Total Sulfur. Once estimated, the remaining samples would be analyzed for Total Sulfur (%) only.

3.15.2 Underground Waste Rock

Underground waste rock will also be sampled at a frequency of one sample per 2,000 tons of waste rock. Since the percentage of waste rock will vary by area, the density of the holes sampled will also need to be adjusted by area. This would result in about 125 samples analyzed for Total Sulfur only (assuming approximately 250,000 tons of underground waste) over the 7 year life of the mine or about 40 samples per year.

3.16 RECORD KEEPING AND REPORTING

Recordkeeping and Reporting. Recordkeeping and reporting will be in accordance with the requirements of DOE Permit No. 8033 and document procedures presented in the hydrological monitoring plan. Appendix J summarizes the detailed water quality monitoring planned. All records of monitoring activities and results shall be retained for the operational life of the mine, should any further actions become necessary. Records and results will be kept in the files at the mine site.

Operational monitoring results will be reported to the Washington DOE and the BLM on a monthly basis. Reporting and filing will be in accordance with the requirements of DOE Permit No. 8033 (Sections S3 and S7). The reports will present the tabulated measurements and the laboratory analytical results. In the case of post-closure monitoring, reports will tally and summarize results from previous efforts; for example quarterly monitoring and reporting would be performed during the first 10 years after mine closure based upon the schedule presented in Appendix J. Thereafter, annual or periodic interpretive monitoring reports will also present a discussion of monitoring data, water quality and quantity trends, and effectiveness of mitigation. Report information will include, but will not be restricted to text, tables and/or maps describing:

- All environmental monitoring data
- Summary of mine development activities
- Summary of surface construction activities
- Summary of reclamation activities
- Summary of operation schedule
- Production records

Figures 46 and 47 show the project schedule and monitoring plan, along with the monitoring evaluation schedule.

REFERENCES

BLM (see U.S. Department of Interior, Bureau of Land Management)

DOE (see Washington Department of Ecology)

Echo Bay Minerals, Inc. 1992a. Plan of Operations, Lamefoot Project.

Echo Bay Minerals, Inc. 1992b. Plan of Operations, Key Project Expansion.

Echo Bay Minerals, Inc. 1993. Plan of Operations April 1993 Modification, Lamefoot Project.

Einarson, D.S., and J.F. Abel, Jr. 1990. "Tunnel Bulkheads for Acid Mine Drainage." Ins Proc. Int'l Symp on Unique Underground Structures. Vol 2. Pp. 71-1. Denver, CO.

Golder Associates. 1994. Draft Report on Backfill Quarry Plan of Operations, Lamefoot Mine. Republic, WA. April.

Hammond, Collier, and Wade-Livingston Associates, Inc. (HCWL). 1993. Backfill Report FOR THE LAMEFOOT MINE. PREPARED FOR ECHO BAY MINERALS **November.**

Holm (Colorado Department of Health). 1989. Personal Communication with R. Moran (W-C).

Marshall, Neil (Echo Bay). 1994. Personal Communication with Jim Johnson (W-C). April 26.

Montana Department of State Lands (MDSL), Montana Department of Health and Environmental Sciences, and the U.S. Forest Service (USFS). 1992. Final Environmental Impact Statement, Stillwater Mine Expansion 2000 TPD: Application to Amend Plan of Operation and Permit No. 00118. September.

Montgomery, James M. **1985.** ~~Page 3-13~~ Water Treatment Principles and Design. John Wiley & Sons, Inc. New York.

Mudder, T.I. (Times Limited). 1994. Technical Memorandum Review of Nitrate Removal Options for the Lamefoot Mine. February 5.

National Coal Board (NCB). 1975. Subsidence Engineers Handbook. NCB Mining Department.

Pommen, L.W. 1983. The Effect on Water Quality of Explosives Use in Surface Mining. Volume 1: Nitrogen Sources, Water Quality, and Prediction and Management of Impacts. MOE Technical Report 4. May.

Soltanpour, P.N., R.H. Follett, and D.G. Westfall. 1985. Guide to Fertilizer Recommendations in Colorado: Soil Analysis and Computer Process. Colorado State University, Cooperative Extension Service.

Unifield Engineering, Inc. and Golder Associates. 1994. Wastewater Nitrate Removal Technology Comparison, Echo Bay Minerals, Lamefoot Mine at Republic, Washington. March 16.

U.S. Forest Service (USFS). 1992. Kettle River Key Project Expansion Final Environmental Impact Statement.

U.S. Department of Interior, Bureau of Land Management (BLM). 1994. Draft Environmental Impact Statement Supplement for the Lamefoot Project. Spokane, WA. February.

U.S. Department of Interior, Bureau of Land Management (BLM). 1992a. Environmental Assessment - Lamefoot Project.

U.S. Department of Interior, Bureau of Land Management (BLM). 1992b. Solid Minerals Reclamation Handbook. BLM Manual Handbook H-3042-1. February.

U.S. Department of Interior, Bureau of Land Management (BLM). 1985. Final Spokane Resource Management Plan/EIS. Spokane District Office. Spokane, WA. August.

Voight B., and W. Pariseau. 1970. "State of the Predictive Art in Subsidence Engineering." Journal Soil Mechanical and Foundation Division. ASCE. Vol. 96. No. 3. SM2. Pp. 721-750.

Washington Department of Ecology. 1988. Final Environmental Impact Statement and Technical Reports, Kettle River Project, Ferry County, Washington. September 23.

Watson, J.T. and A.J. Danzig. 1993. Pilot-Scale Nitrification Studies Using Vertical-Flow and Shallow Horizontal-Flow Constructed Wetland Cells. Chap. 31. CRC Press.

TABLES

TABLE 1

COUNTY PERMITS REQUIRED FOR THE KETTLE RIVER OPERATIONS

COUNTY PERMITS REQUIRED FOR THE KETTLE RIVER OPERATION								
AGENCY	TYPE OF APPROVAL	CITATION	DATE OF APPROVAL					
			Kettle River Mine	Overlook Mine	Key East & West Mines	Key Mill	Lamefoot Mine	K-2 Site[1]
Ferry County								
	Building Permit		09/15/87	08/30/88	--	08/30/88		--
	Road Relocation and/or Access		06/20/88		--	08/30/88	01/11/93	--
	Ferry County Mining Ordinance	RCW 36.32.120	--	05/14/92	05/14/92	N/A	05/04/92	5/14/92
	On-Site Sewage (Tri-County Health Department)	248.96 WAC	10/09/87	10/21/88	--	08/26/88	07/29/92	--

[1] Permits required will be determined based on potential plans for the K-2 deposit to be determined late 1993 or 1994.

TABLE 2

STATE PERMITS REQUIRED FOR THE KETTLE RIVER OPERATIONS

STATE PERMITS OR APPROVALS REQUIRED FOR THE KETTLE RIVER PROJECT								
AGENCY	TYPE OF APPROVAL	CITATION	DATE OF APPROVAL					
			Kettle River Mine	Overlook Mine	Key East & West Mines	Key Mill	Lamefoot Mine	K-2 Site[1]
Department of Ecology								
	Burning Permit Change of Place or Purpose of Use Approval	RCW 70.94.650; RCW 90.03.300	6/07/91	6/07/91	--	6/07/91	1/11/93	--
	SEPA Compliance (production)	197-11 WAC	9/22/87	9/23/88	8/1/92	9/23/88	Pending	--
	Reservoir Permit and Dam Safety Approval		--	--	--	4/89	--	--
	Waste Discharge Permit	RCW 90.48.160	12/8/89	12/8/89	10/23/92	12/8/89	10/23/92	--
	Change of Purpose of Surface Water Right		--	--	--	8/31/88	--	--
	New Source Construction Approval Public Groundwater Withdrawal Permit	RCW 70.95.152; RCW 90.44.050	4/24/89	4/24/89	Pending	3/1/89	Pending	--
	Sewage Facilities Approval	RCW 90.48.110	--	--	--	--	--	--
	Air Pollution Source Regulation (production)	RCW 70.94	9/8/89	9/8/89	9/2/92	9/8/89	Pending	--

TABLE 2
(Concluded)

STATE PERMITS OR APPROVALS REQUIRED FOR THE KETTLE RIVER PROJECT								
AGENCY	TYPE OF APPROVAL	CITATION	DATE OF APPROVAL					
			Kettle River Mine	Overlook Mine	Key East & West Mines	Key Mill	Lamefoot Mine	K-2 Site[1]
	NPDES Permit	90448 RCW 9052 RCW WAC 173-220					4/9/93	
Department of Natural Resources								
	Burning Permit	RCW 76.04.150; RCW 76.04.170	6/12/91	6/12/91	10/26/92	6/12/91	5/19/92	--
	Surface Mine Reclamation Permit	RCW 78.44	--	--	10/6/92	--	--	--
Department of Social Health Services (DSHS)								
	On-Site Sewage	248-96 WAC	10/9/87	10/21/88	--	8/26/88	7/29/92	--

[1] Permits required will be determined based on potential plans for the K-2 deposit to be determined late 1993 or 1994.

TABLE 3

FEDERAL PERMITS REQUIRED FOR THE KETTLE RIVER OPERATIONS

FEDERAL PERMITS REQUIRED FOR THE KETTLE RIVER PROJECT								
AGENCY	TYPE OF APPROVAL	CITATION	DATE OF APPROVAL					
			Kettle River Mine	Overlook /Key Mine	Key East & West Mines	Key Mill	Lamefoot Mine	K-2 Site
Forest Service								
	Plan of Operations for Exploration and/or Mining	36 CFR 228	--	8/27/92	8/27/92	--	--	--
	SEPA/NEPA Compliance		--	8/27/92	8/27/92	--	--	--
BLM								
	Plan of Operation Approval	43 CFR 3809	--	9/30/92	9/30/92	--	7/28/92	--
	NEPA Compliance		--	9/30/92	9/30/92	--	7/28/92	--

TABLE 4

MAXIMUM VOID HEIGHT FOR PLANNED UPPER LAMEFOOT STOPES AFTER BACKFILLING

Stope (Zone)	Section	Crown Pillar Thickness	Maximum Allowable Void Height
2	N26000	83 feet	22.8 feet
	N26100	99 feet	27.2 feet
	N26200	114 feet	31.4 feet
	N26300	113 feet	31.1 feet
	N26400	113 feet	31.1 feet
3	N26500	64 feet	17.6 feet
	N26600	54 feet	14.9 feet
	N26700	50 feet	13.8 feet
	N26800	56 feet	15.4 feet
	N26900	50 feet	13.8 feet
	N27000	50 feet	13.8 feet
4	N27200	42 feet	11.6 feet
	N27300	100 feet	27.5 feet
	N27400	104 feet	28.6 feet
	N27500	150 feet	41.3 feet

TABLE 5

CROWN PILLAR WIDTH AND THICKNESSES FOR UPPER STOPES

Stope	Section	Crown Pillar Width (ft)	Crown Pillar Thickness (ft)	Comments
2	26000N	26	83	Closest to surface
	26100N	37	99	
	26200N	44	114	Widest stope section
	26300N	37	113	
	26400N	40	113	
3	26500N	65	64	Widest stope section
	26600N	23	54	
	26700N	26	50	
	26800N	16	56	
	26900N	16	50	Closest to surface
	27000N	16	50	Closest to surface
Pillar	27100N			
4	27200N	16	42	Closest to surface
	27300N	55	100	
	27400N	54	104	
	27500N	64	150	Widest stope section

TABLE 6

MAXIMUM POSSIBLE SURFACE SUBSIDENCE (INCHES) ABOVE NEAR-SURFACE STOPES[1]

Section[2]	Stope[2] Number	Open Stopes No Backfill	Low-modulus Backfill	High-modulus Backfill
60-foot stope width				
N26090	2	0.95	0.94	0.83
N26930	3	1.35	1.32	1.13
N27250	4	1.37	1.34	1.15
130-foot stope width				
N26090	2	1.24	1.23	1.15
N26930	3	1.68	1.67	1.52
N27250	4	1.62	1.61	1.42

[1] Conservatively assumed equal to total vein closure and no failure of the surface crown pillar.

[2] Section and stope number from Plan of Operations (Echo Bay 1993), Figures 10A and 10B.

TABLE 7

MAXIMUM POSSIBLE SURFACE SUBSIDENCE-INDUCED STRAINS ABOVE NEAR-SURFACE STOPES[1]

Section[2]	Stope[2] Number	Open Stopes No Backfill	Low-modulus Backfill	High-modulus Backfill
Maximum Tensile Strain ($\nu\epsilon$) 60-foot Stope Width				
N26090	2	1030	1020	900
N26930	3	1460	1430	1220
N27250	4	1500	1460	1250
130-foot Stope Width				
N26090	2	1340	1330	1250
N26930	3	1820	1810	1650
N27250	4	1760	1740	1540
Maximum Compressive Strain ($\nu\epsilon$) 60-foot Stope Width				
N26090	2	870	860	760
N26930	3	1240	1210	1040
N27250	4	1270	1230	1050
130-foot Stope Width				
N26090	2	1050	1050	980
N26930	3	1430	1420	1290
N27250	4	1380	1370	1210

[1] In $\nu\epsilon$ - micro-strain, millionths inch per inch.

[2] Section and stope number from Plan of Operations (Echo Bay 1993), Figures 10A and 10B.

TABLE 8

LAMEFOOT PROJECT COMPONENT EQUIPMENT LIST

EQUIPMENT	LAMEFOOT PROJECT COMPONENTS NUMBER OF PIECES OF EQUIPMENT AT EACH SITE	
	Lamefoot Mine	
	1993	1994
Front End Loader	1	2
Pit Truck (50 ton capacity for waste rock)		
30 or 15 Ton Underground Truck	4	7
Haul Truck (30 ton capacity for ore)	4	9
Haul Truck (20 ton capacity for ore)	0	0
Powder Truck	1	1
Dozer (D-9 or equivalent)		
Dozer (D-8 or D-6 or D-4)	1	1
Water Truck (0000 gallon)		
Combination Water and Drill Pipe Truck (3000 gallon)	3	0
Supervisory Vehicles (pickups)	3	3
Portable Flood Lights		
Rotary Blasthole Drills	0	1

TABLE 8
(Concluded)

EQUIPMENT	LAMEFOOT PROJECT COMPONENTS NUMBER OF PIECES OF EQUIPMENT AT EACH SITE	
	Lamefoot Mine	
	1993	1994
Truck-mounted Reverse Circulation Rotary Drill	1	0
Track-mounted Reverse Circulation Rotary Drill	1	0
Truck-mounted Diamond Core Drill	1	2
Skid-mounted Diamond Core Drill	2	0
Crane	1	1
Drill Jumbo	2	2
6 or 3 yard scoops	4	4
Welding Truck	1	0
Air Truck	1	0
Compressor	2	2
Generator	2	0
Scissor-Lift	2	2
Excavator	1	0
Graders	0	1
Tractor	0	2
Trammer	0	2

TABLE 9

LAMEFOOT OPERATIONS
FUEL, OIL, LUBRICANTS AND CHEMICALS - 1993 CONSUMPTION DATA

		Number of Deliveries Per Month	Quantity Per Delivery	Quantity Used Per Month	Quantity Stored At Mill
Fuel		8	500 gal.	4,000 gal.	500 gal.*
Oil & Lubricants		2	5,000 qts.	10,000 qts.	10,000qts.
Explosives		2	15,000 lb.	25,000-30,000 lb.	40,000*
Chemicals	Sodium Cyanide	4	57,000 lb.	230,000 lb.	75,000 lb.
	Calcium Hydroxide	12	54,000 lb.	650,000 lb.	100,000 lb.
	Acid (nitric)	0.3	45,000 lb.	15,000 lb.	30,000 lb.
	Other Propane	0.5	9,000 gal.	4,500 gal.	20,000 gal.
	Oxygen	4	42,000 lb.	16,000 lb.	150,000 lb.
	SO2	11	54,000 lb.	550,000 lb.	80,000 lb.
	Caustic	2	45,000 lb.	90,000 lb.	30,000 lb.

* Stored at Lamefoot Mine Site.

TABLE 10

PROJECT RELATED WATER PRODUCTION AND USE

PROJECT-RELATED WATER PRODUCTION AND USE				
Kettle River Project Component	Water Production and Use			
	Average Daily Consumption During Operations (gallons)	Consumption During Startup (gallons per minute)	Production Rate from Mine (gallons per minute)	On-site Storage Capacity (gallons)
Kettle River Mine	1,000 (from years 1988-1992)	N/A	0 to 15 Current: 0	2,000
Overlook Mine	2,000 (from years 1988-1992)	N/A	0 to 10 Current: 0	0
Key East & West Mines	5,000	N/A	0 to 50	5,000
Lamefoot Mine Supply Well	2,000	N/A	0 to 50	0
Key Mill and Tailings Facility	70,000	600-700 gpm	N/A	5,900
K-2 Exploration	N/A	N/A	N/A	0

TABLE 11

ESTIMATED AREA OF SURFACE DISTURBANCE FOR KETTLE RIVER PROJECT COMPONENTS

Kettle River Project Component	Estimated Area (acres) of Disturbance				
	Privately Owned	State Owned	Federal - administered by Forest Service	Federal - administered by BLM	Total
Underground Mines and Associated Facilities					
Kettle River	25	0	0	0	25
Overlook	22	0	3	5	30
Lamefoot (and associated exploration)	38	0	0	7-12	45-50
Surface Mines and Associated Facilities					
Key East & West Mines	13	0	7	0	20
Key E/W Waste Rock, Soil Stockpiles, Roads, etc.	49	0	12	0	61
Transportation Routes					
Key East/West to Overlook Mine	29	0	0	2	31
Overlook Mine to Key Mill	23	0	0	0	23
Lamefoot to Key Mill♦ (excluding public roads)	0	0	0	0	0
Exploration Sites					
K-2	15	0	0	0	15
Powerlines and Substations					
Kettle River Mine♦	0	0	0	0	0
Lamefoot Mine♦	0	0	0	0	0
Overlook Mine	4	0	0	0	4
Key East & West Mines	0	0	0	0	0

TABLE 11
(Concluded)

Kettle River Project Component	Estimated Area (acres) of Disturbance				
	Privately Owned	State Owned	Federal - administered by Forest Service	Federal - administered by BLM	Total
Underground Mines and Associated Facilities					
Ore Processing Facilities					
Key Mill	30	0	0	0	30
Tailings Facility	125	0	0	0	125
Total Estimated Acres Disturbed					274-279

♦ Areas Included in Site Facility.

TABLE 12

COVERSOIL SUMMARY

Location	Available Ft^3	Required Ft^3
Sidecast and Small Stockpiles in Areas A,B,C,D,E,F AND G	104,355	104,355
Area H	0	53,550
Surface Facilities	235,000	181,450
Total	339,400	339,400

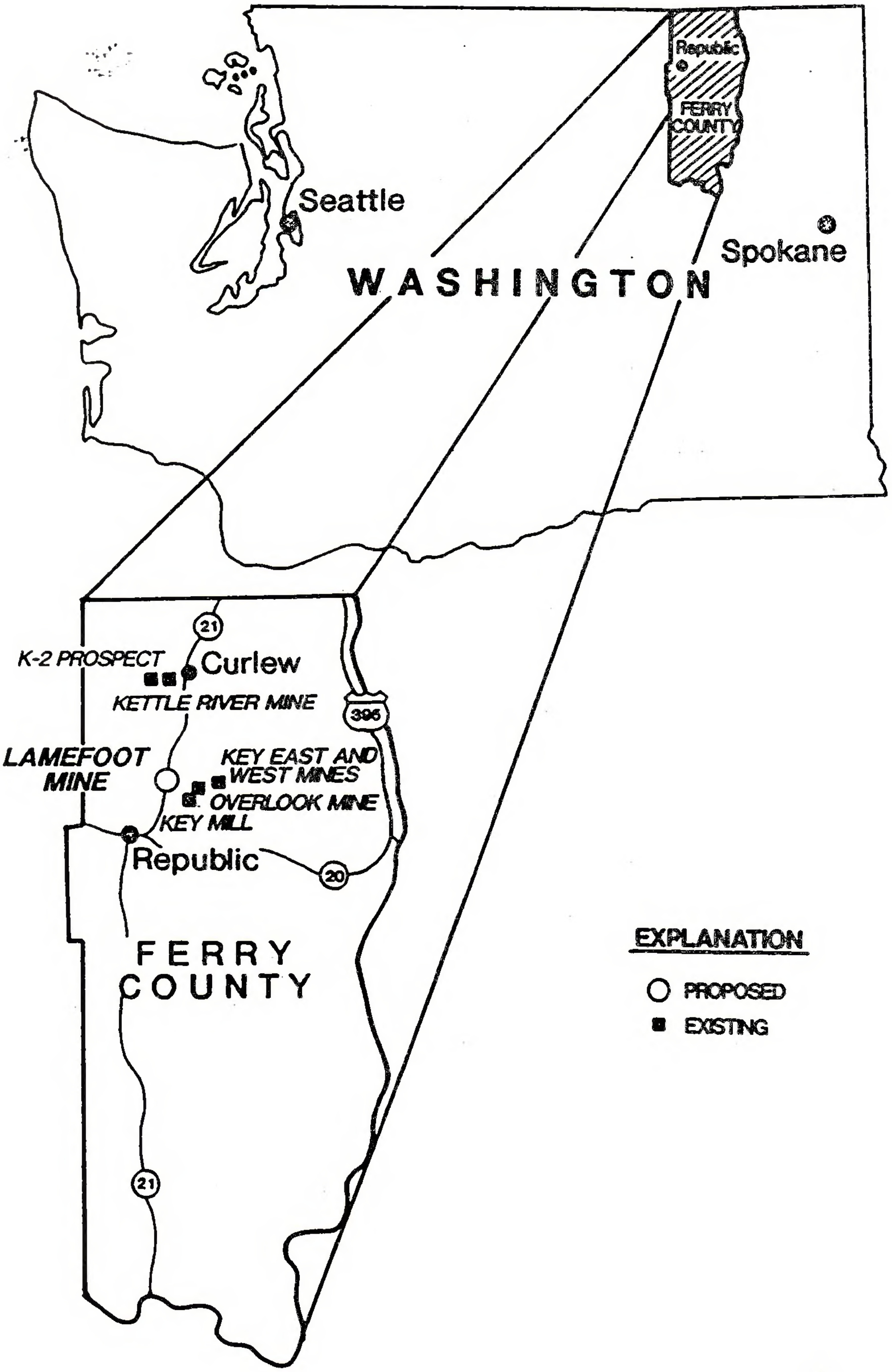
Note: Areas are shown on various figures in Lamefoot 1992.

TABLE 13

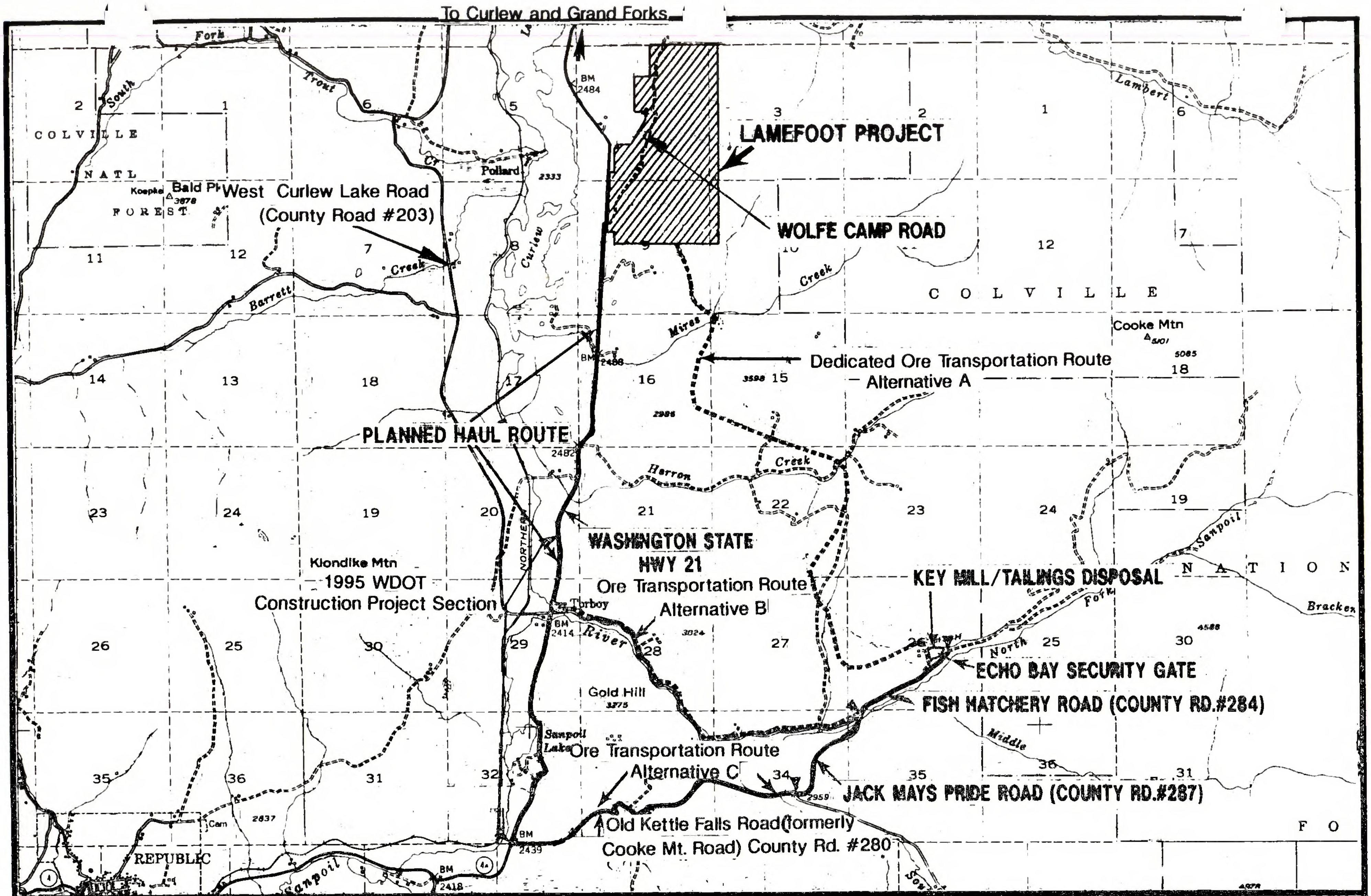
FERRY COUNTY NOXIOUS WEED LIST

"B" Designate		
Tansy Ragwort	Scotch Thistle	Yellow Starthistle
Leafy Spurge	Rush Skeletonweed	Spotted Knapweed
Scotch Broom	Common Bugloss	Common Catscar
Oxeye Daisy	Orange Hawkweed	Yellow Hawkweed
Black Knapweed	Brown Knapweed	Meadow Knapweed
Plumeless Thistle		
Russian Knapweed (in north half of country)	Diffuse Knapweed is a "B" designate on vehicle corridors, buffer strips, and in areas of limited distribution.	
Class "B" or "C"		
Field Blindweed	Eurasian Watermilfoil	Houndstounge
Knapweed Complex	St. Jonswort	Canada Thistle
Yellow Toadflax	Dalmation Toadflax	Black Henbane
Poison Hemlock	Baby's Breath	Absinth Wormwood
Western Waterhemlock	Whitetop (Hoary Cress)	Jointed Goatgrass
Musk Thistle	Puncturevine	

FIGURES

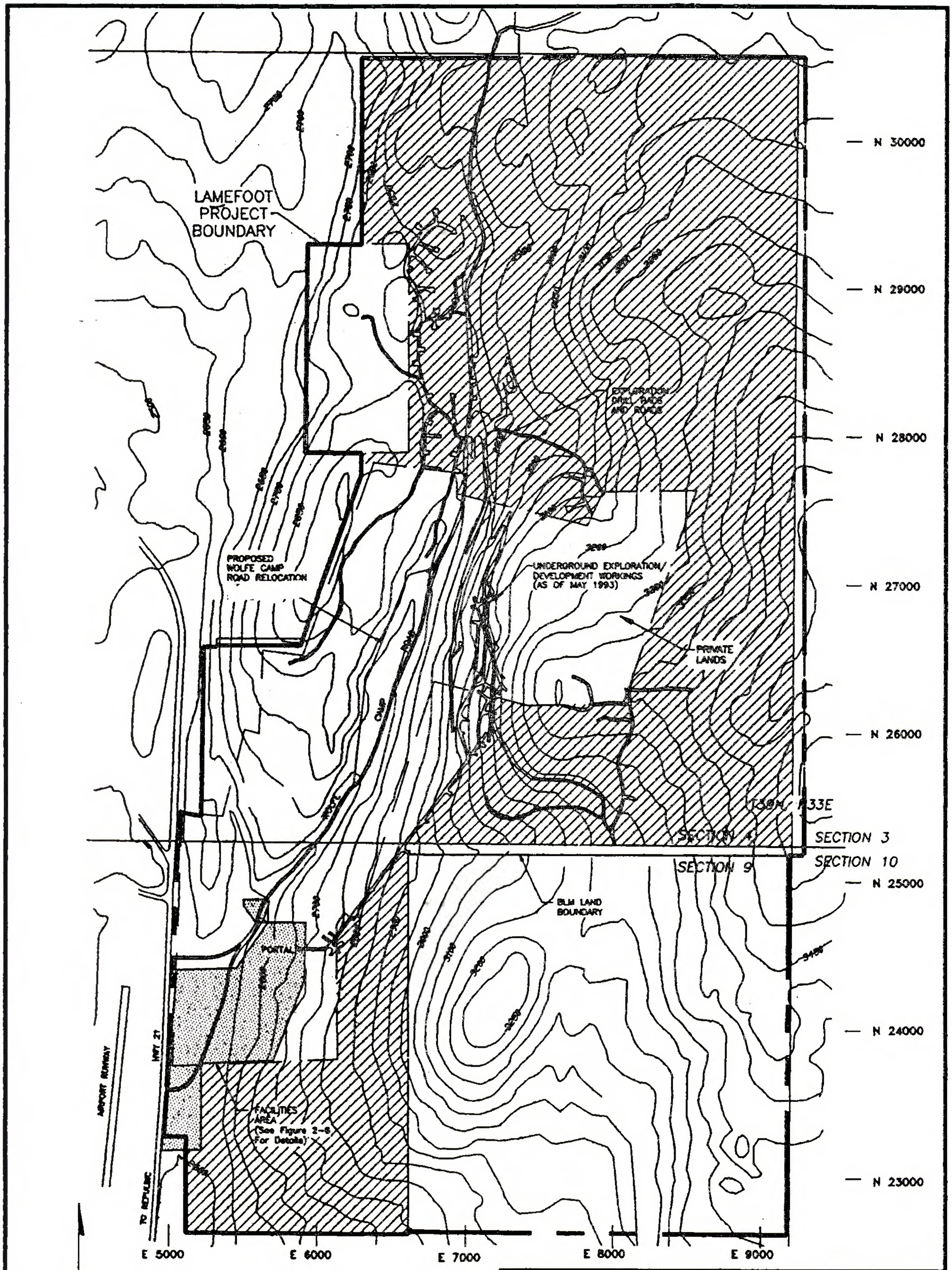


CLIENT/PROJECT	ECHO BAY MINERALS Co. LAMEFOOT MINE KETTLE RIVER OPERATIONS REPUBLIC, WASHINGTON	TITLE	GENERAL LOCATION MAP SOURCE: ECHO BAY, 1993a		
DRAWN	CHECKED J.C.	REVIEWED L.E.C.	DRAWN BDL	DATE APRIL 1993	JOB NO. 933-2614
			CHECKED LEC	SCALE NA	DWG NO./REV. NO.
			REVIEWED SHM	FILE NO.	FIGURE NO. 1



CLIENT/PROJECT		TITLE
ECHO BAY MINERALS Co. LAMEFOOT MINE KETTLE RIVER OPERATIONS REPUBLIC,WASHINGTON	ECHO BAY MINERALS, INC.	LAMEFOOT PROJECT SITE ACCESS AND REGIONAL TOPOGRAPHY Source: Echo Bay 1993a

DRAWN	CHECKED	REVIEWED	DATE	SCALE	FILE NO.	JOB NO.	DWG NO./REV.NO.	FIGURE
BDL	LEC	SHM	APRIL 1993	1"= 1 MILE		933-2614		2



ECHO BAY LAMEFOOT PROJECT

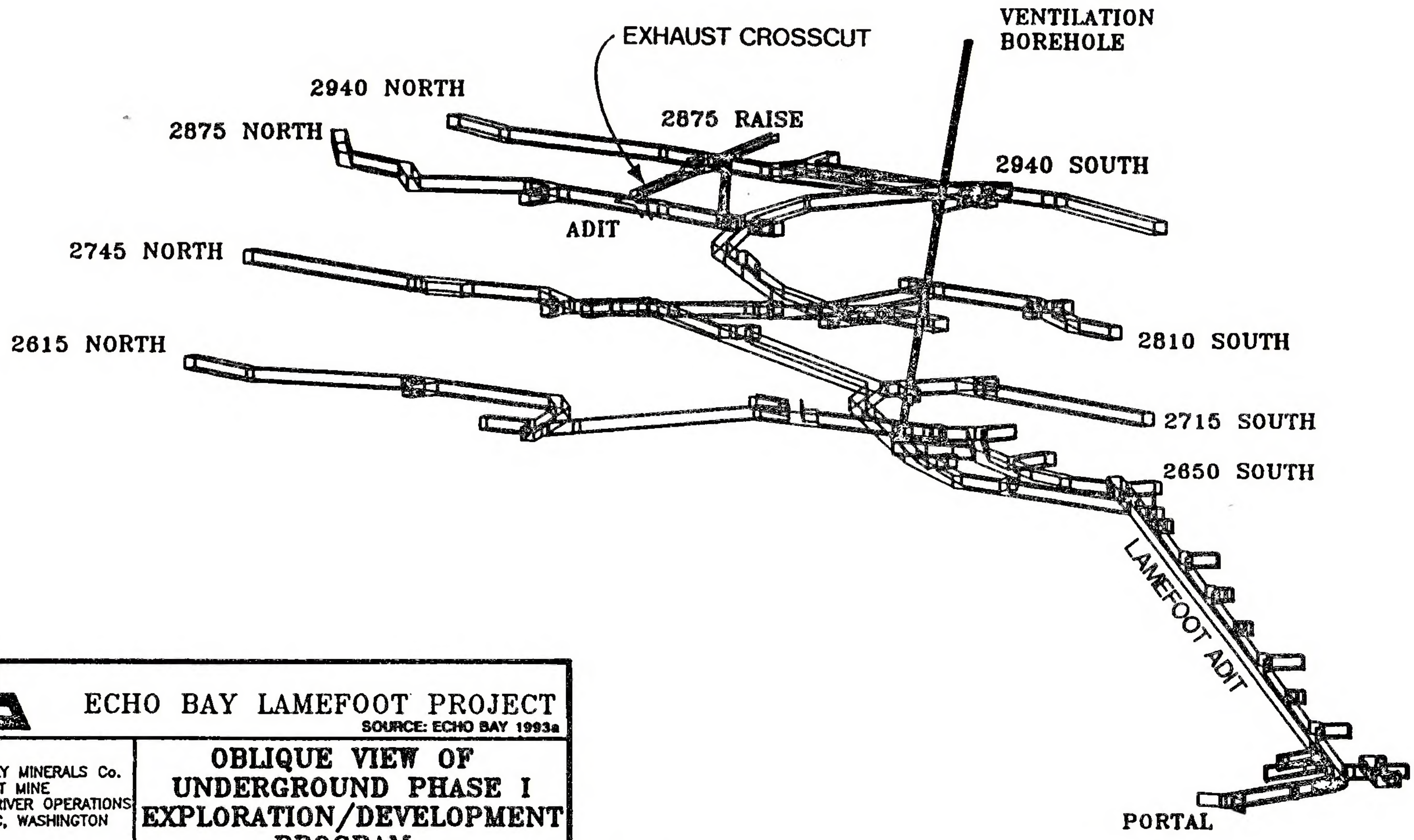
SOURCE: ECHO BAY 1993a

ECHO BAY MINERALS Co.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

SITE TOPOGRAPHY SHOWING EXISTING DRILL ROADS, FACILITIES AND EXPLORATION/ DEVELOPMENT WORKINGS

DATE	SCALE 1" = 600'	FIG. NO. 3

ELEVATED VIEW LOOKING NORTHEAST



ECHO BAY LAMEFOOT PROJECT

SOURCE: ECHO BAY 1993a

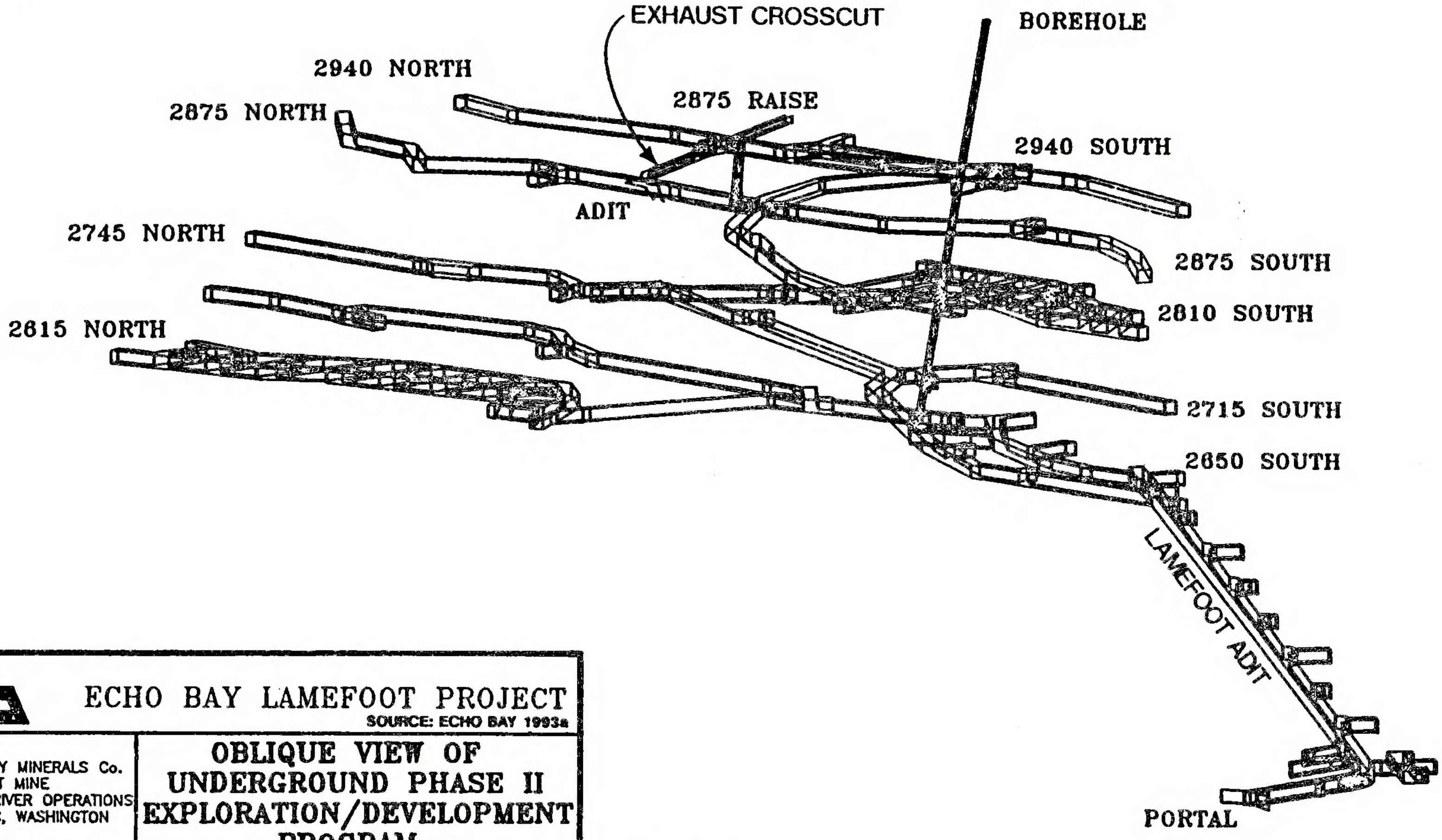
ECHO BAY MINERALS Co.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

OBLIQUE VIEW OF UNDERGROUND PHASE I EXPLORATION/DEVELOPMENT PROGRAM

DATE APRIL 93	SCALE	FIG. NO. 4

FILE: OBLPHI.DWG

ELEVATED VIEW LOOKING NORTHEAST



ECHO BAY LAMEFOOT PROJECT

SOURCE: ECHO BAY 1993a

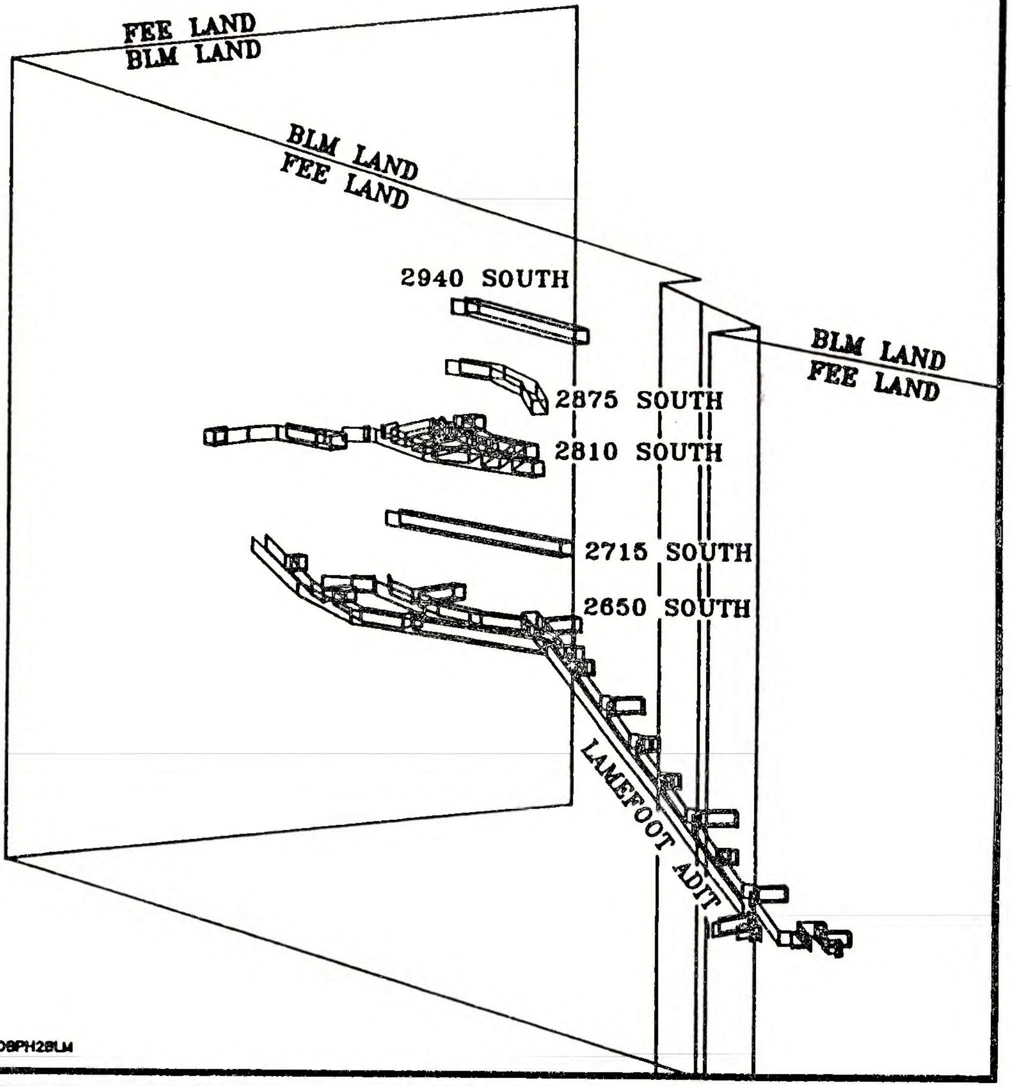
ECHO BAY MINERALS Co.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

OBLIQUE VIEW OF UNDERGROUND PHASE II EXPLORATION/DEVELOPMENT PROGRAM

DATE APRIL 93	SCALE	FIG. NO. 5

FILE: OBLPHII.DWG

ELEVATED VIEW LOOKING NORTHEAST



ECHO BAY LAMEFOOT PROJECT

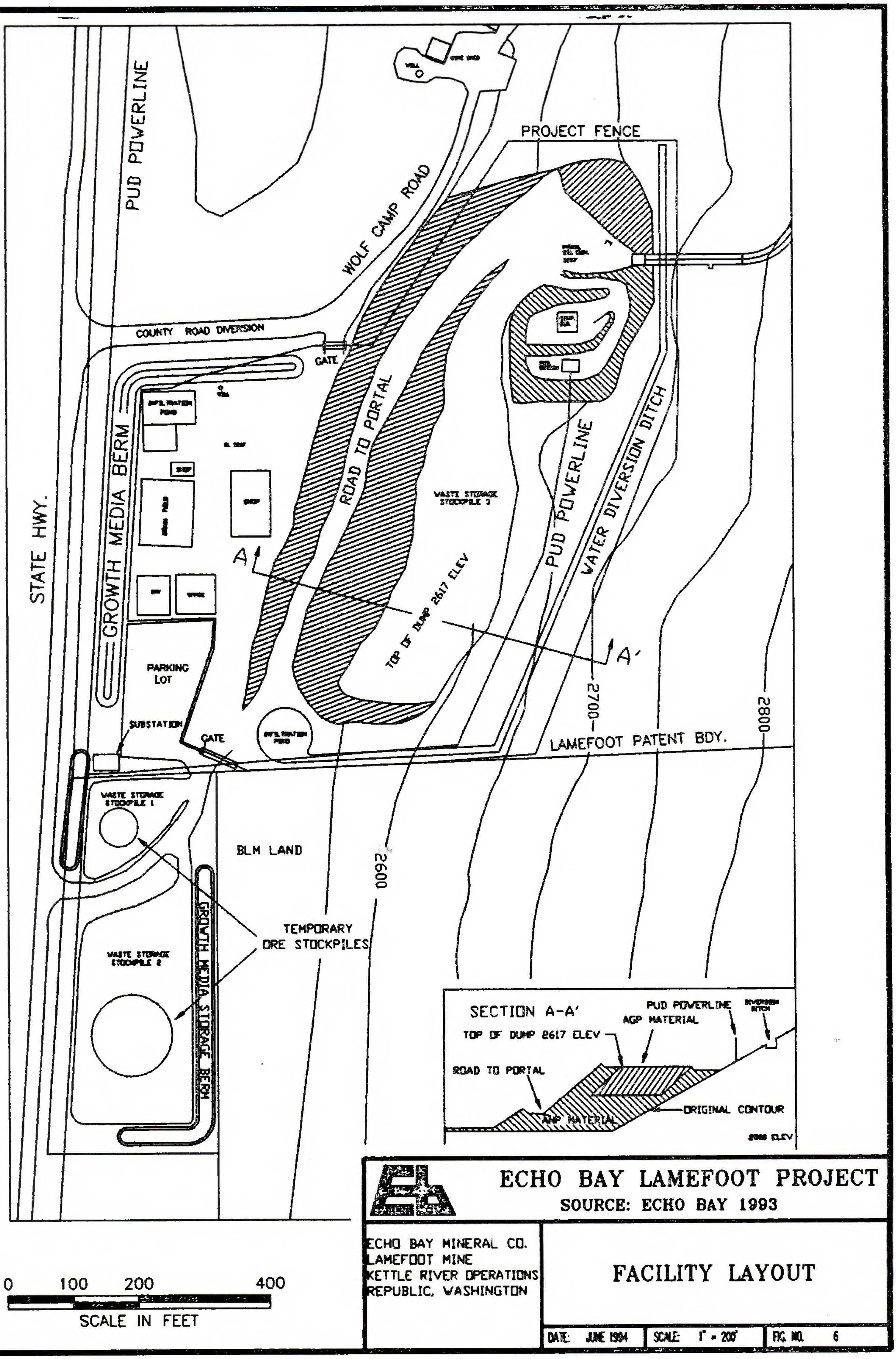
SOURCE: ECHO BAY 1993a

ECHO BAY MINERALS Co.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

OBLIQUE VIEW OF UNDERGROUND PHASE II PROGRAM ON BLM LAND

DATE APRIL 93 | SCALE | FIG. NO. 5A

FILE: 08PH2BLM



ECHO BAY LAMEFOOT PROJECT

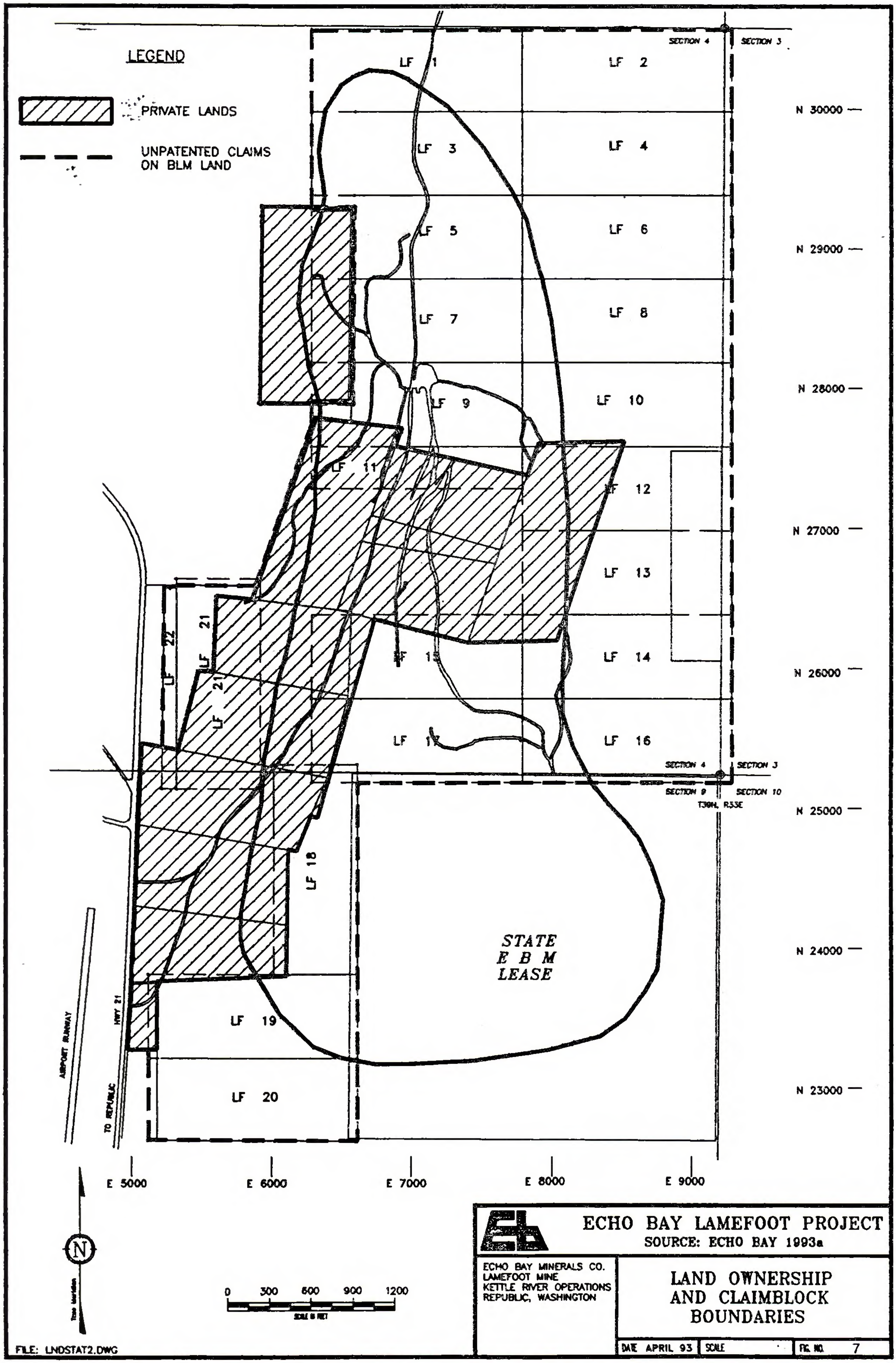
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

FACILITY LAYOUT

DATE: JUNE 1994 | SCALE: 1" = 200' | FIG. NO. 6

LFDUMP2



ECHO BAY LAMEFOOT PROJECT

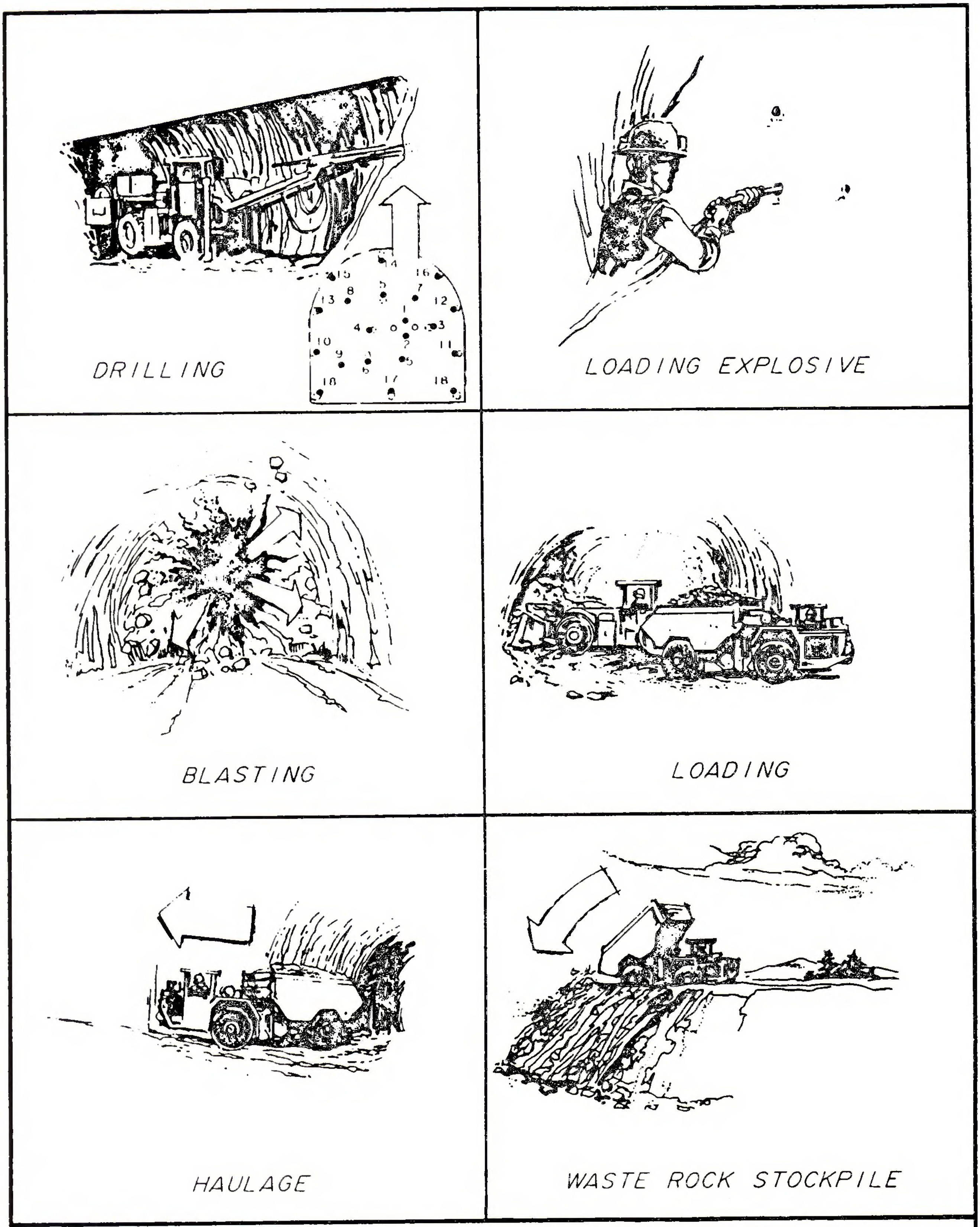
SOURCE: ECHO BAY 1993a

ECHO BAY MINERALS CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

LAND OWNERSHIP AND CLAIMBLOCK BOUNDARIES

DATE APRIL 93 | SCALE | FIG. NO. 7

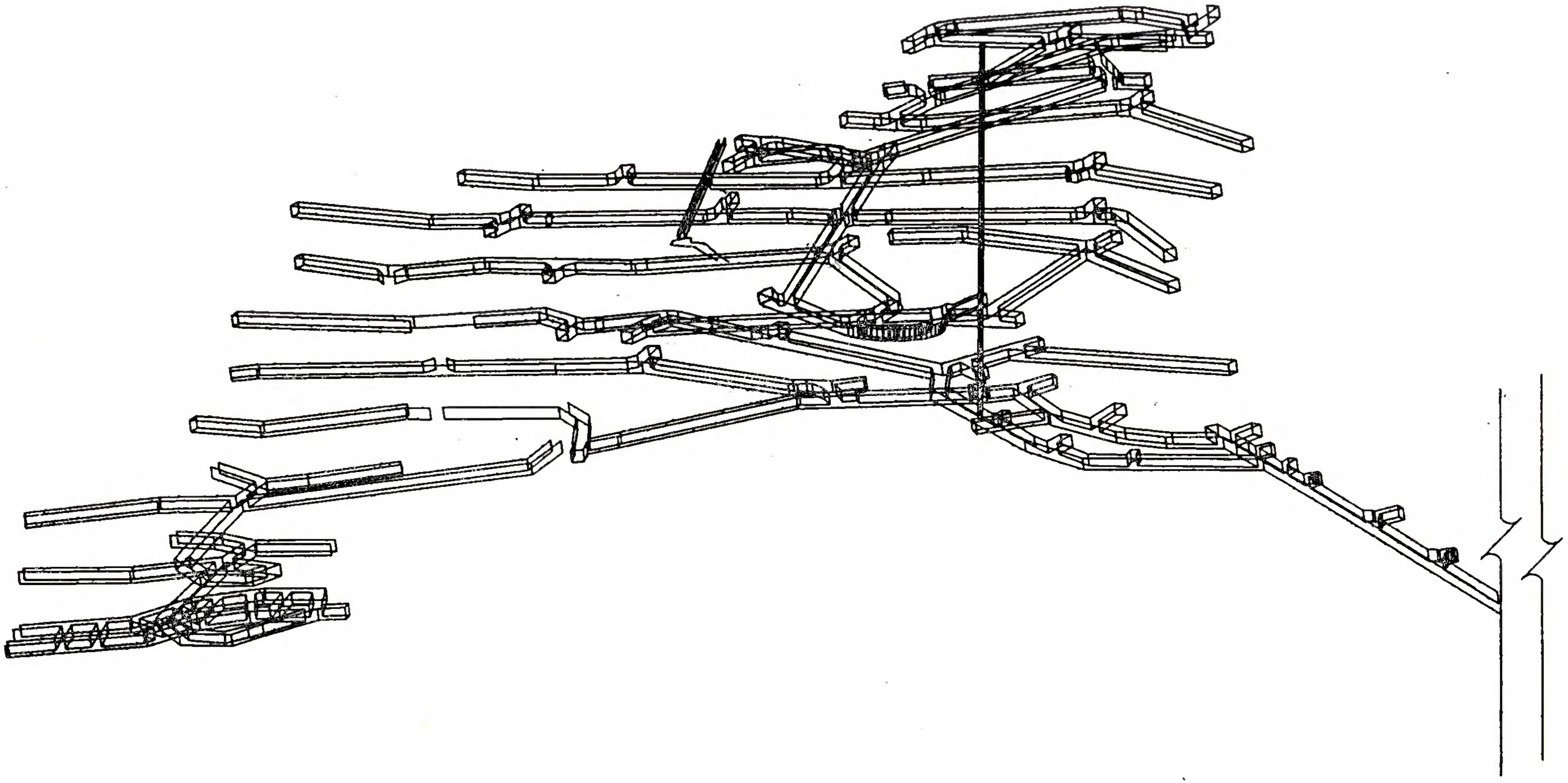
FILE: LNDSTAT2.DWG



MINE DECLINE DEVELOPMENT TECHNIQUES

FIGURE 8

SOURCE: STATE OF WASHINGTON DOE 1988



FILE: 2-3

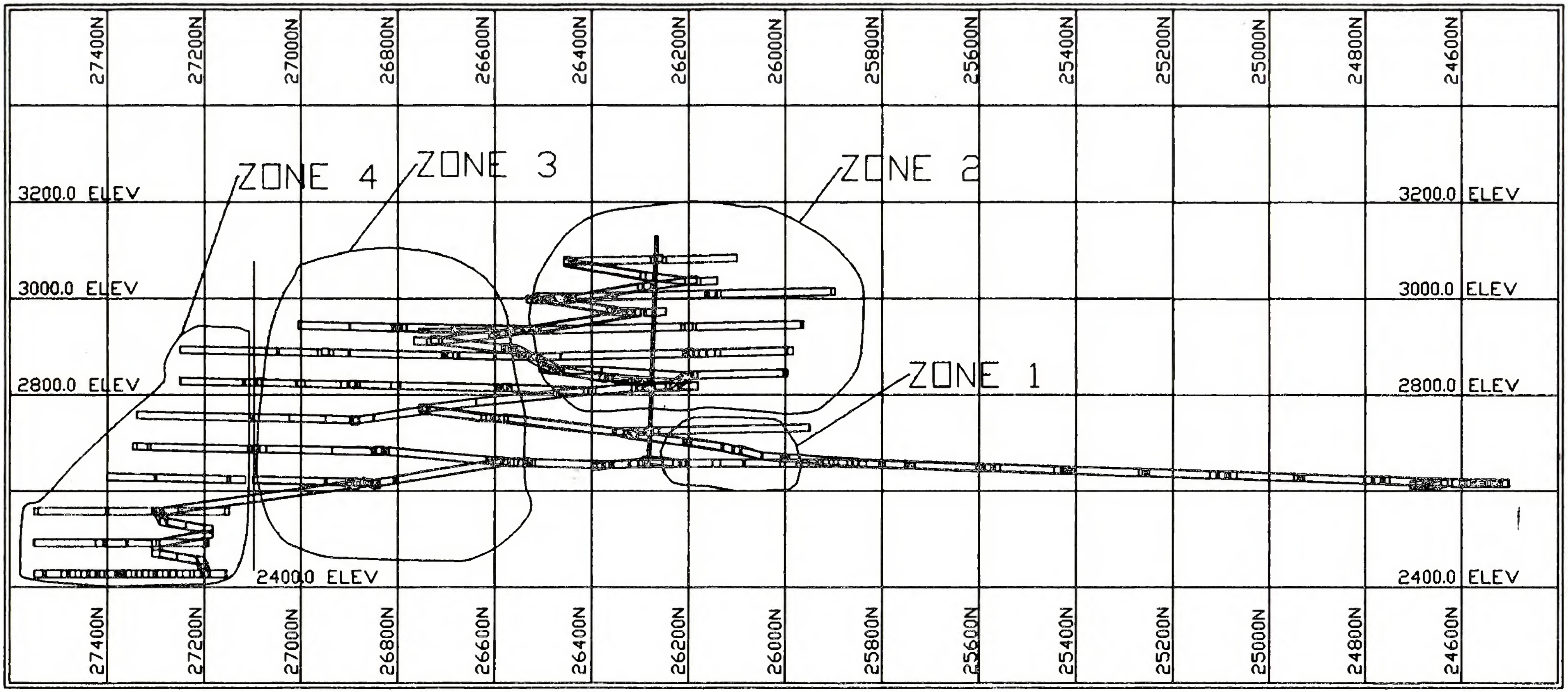
ECHO BAY LAMEFOOT PROJECT

Source: Adapted from Echo Bay 1993a

ECHO BAY MINERALS Co.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

OBLIQUE VIEW OF UNDERGROUND EXPLORATION AND COMMERCIAL DEVELOPMENT PROGRAM

NOV 93 | SCALE 1"=100' | FIG. NO. 9



ECHO BAY LAMEFOOT PROJECT

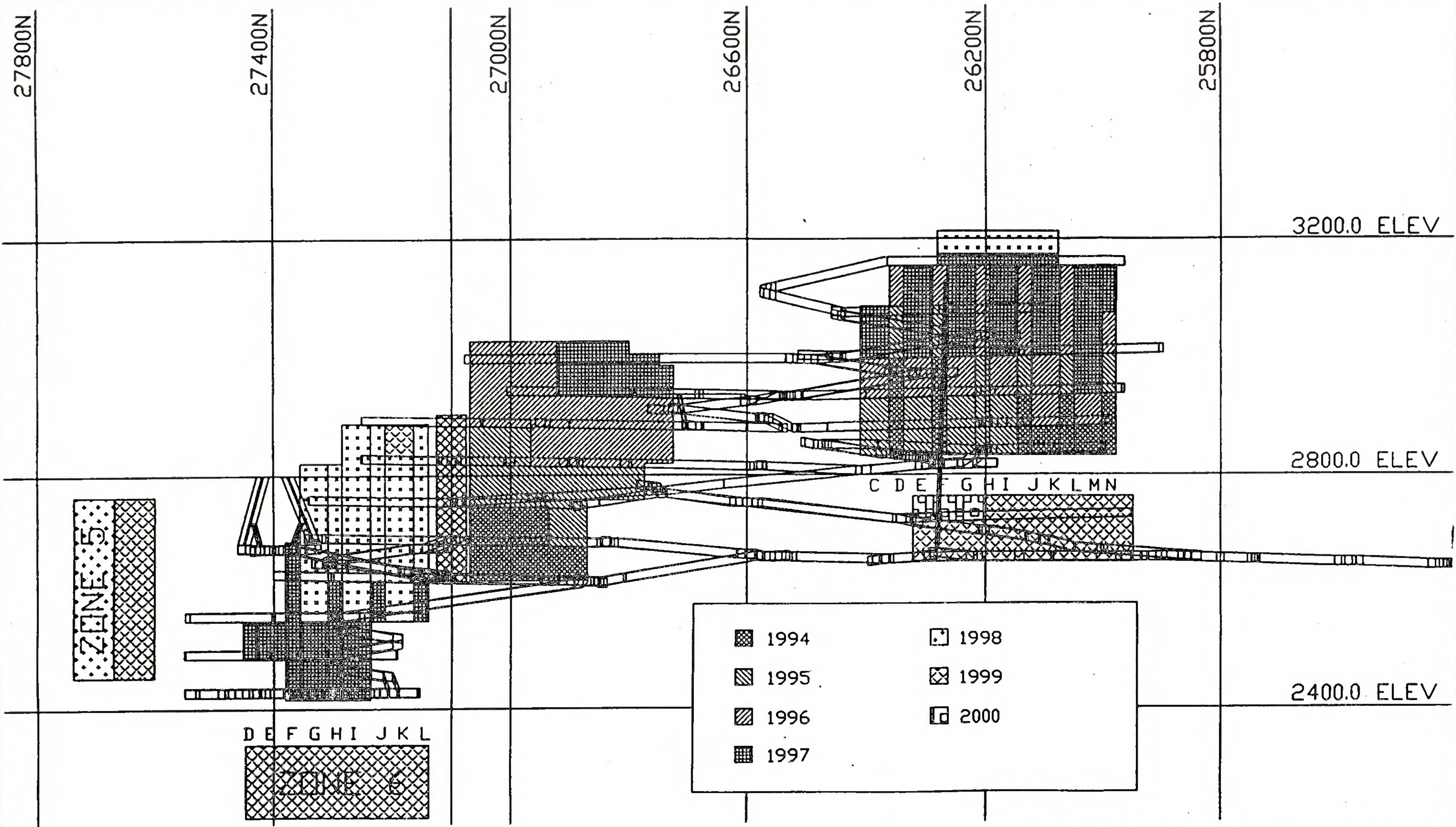
Source: Golder 1993

ECHO BAY MINERALS Co.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

MINING SEQUENCE
LAMEFOOT PROJECT
LONG SECTION LOOKING EAST

FILE: 2-9A

DATE: NOV 19,93 | SCALE: | FIG. NO. 10



ECHO BAY LAMEFOOT PROJECT

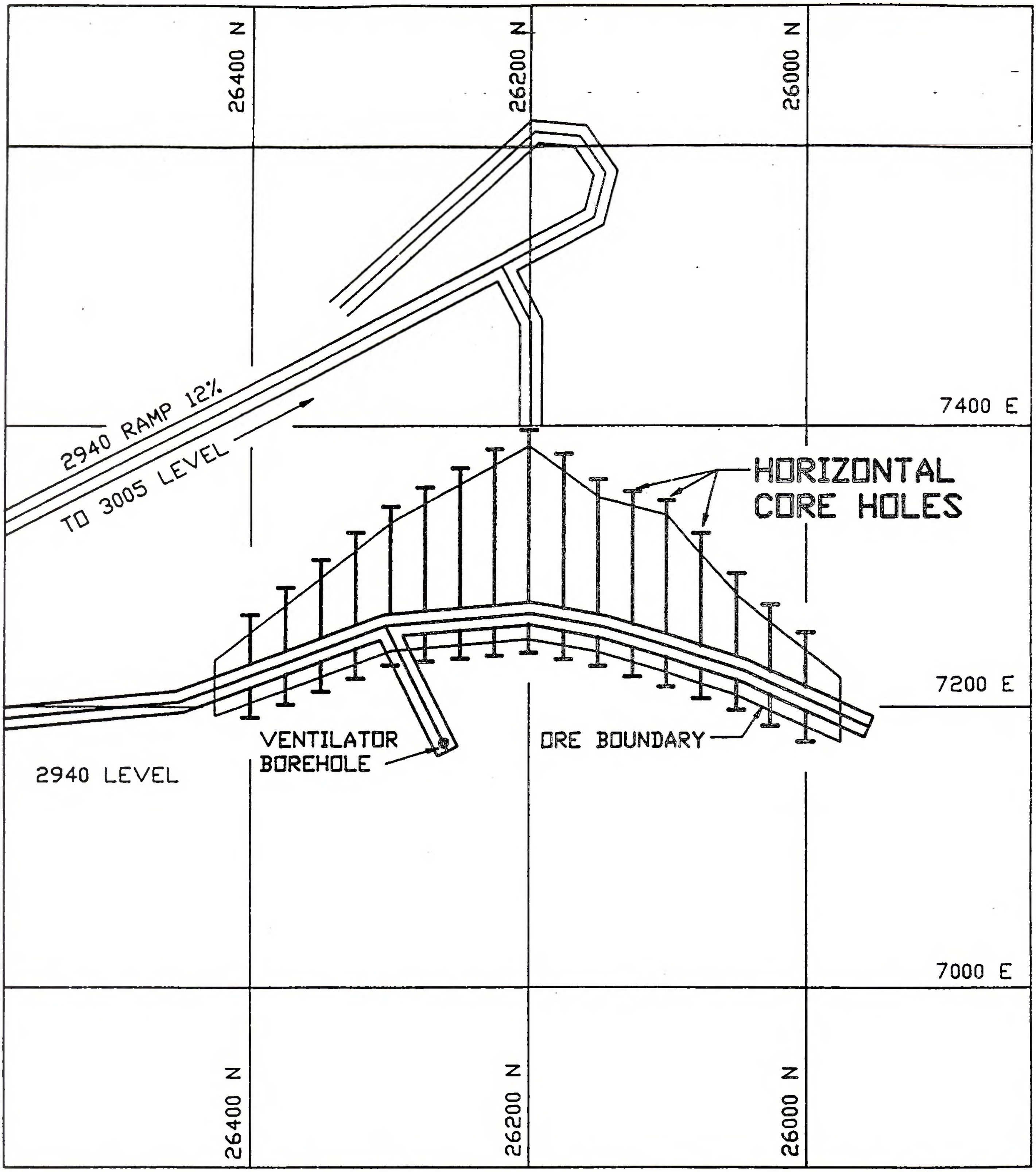
SOURCE: ECHO BAY 1993

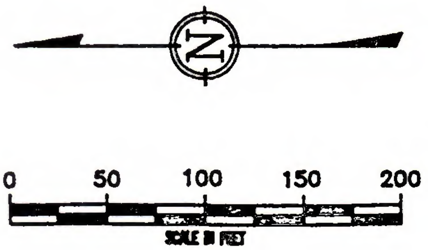
ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

LAMEFOOT PROJECT LONG SECTION LOOKING EAST

DATE: MAY 1994 SCALE: FIG. NO. 11

LFSECE





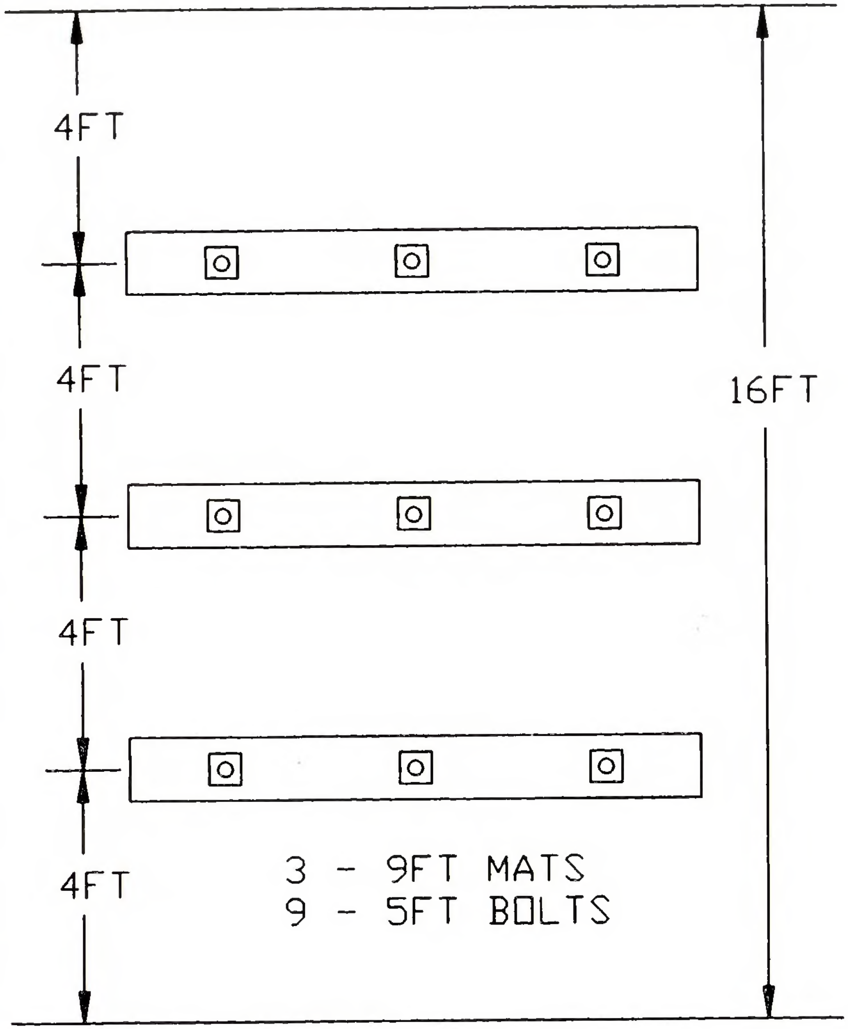
ECHO BAY LAMEFOOT PROJECT

COMPILED	DATE	DRAFTED	DATE
SKIPPER	11-91		
REVISED	DATE	REVISED	DATE

PLAN VIEW OF TYPICAL LAYOUT OF HOLES DRILLED IN SUBLEVEL DRIFTS

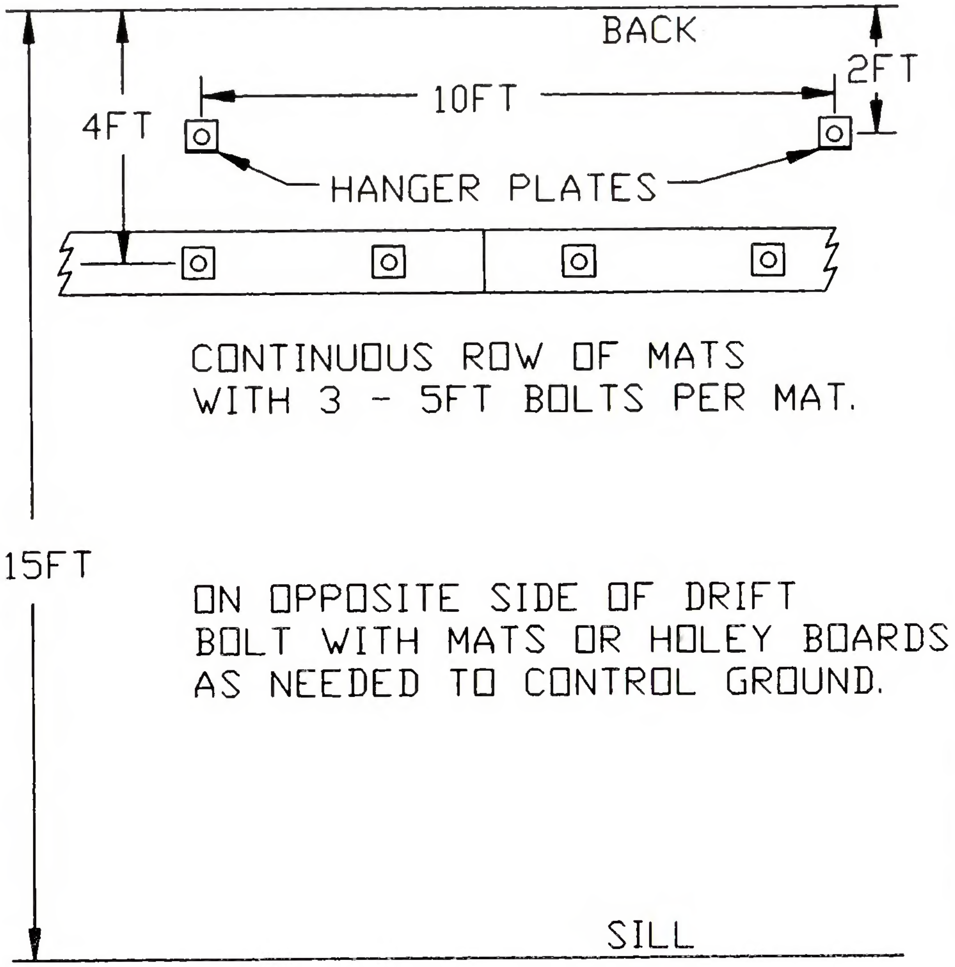
DATE APRIL 93
SCALE 1"=100'
FIG. NO. 12

FILE: BAZOOKA4.DWG



3 - 9FT MATS
9 - 5FT BOLTS

BACK VIEW



CONTINUOUS ROW OF MATS
WITH 3 - 5FT BOLTS PER MAT.

ON OPPOSITE SIDE OF DRIFT
BOLT WITH MATS OR HOLEY BOARDS
AS NEEDED TO CONTROL GROUND.

UTILITIES SIDE OF DRIFT

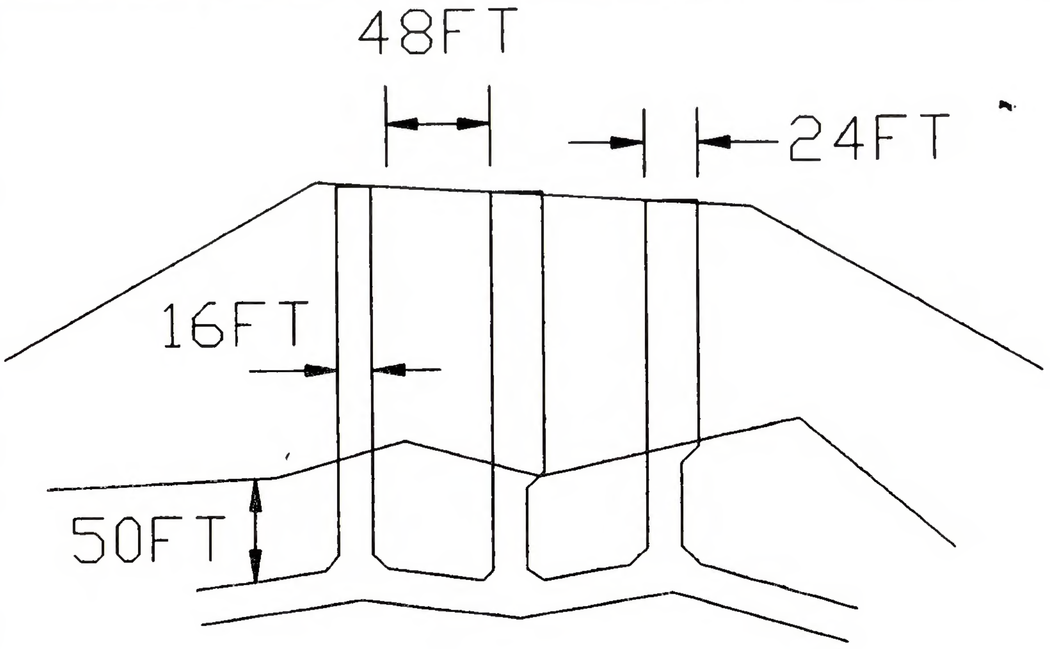
ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

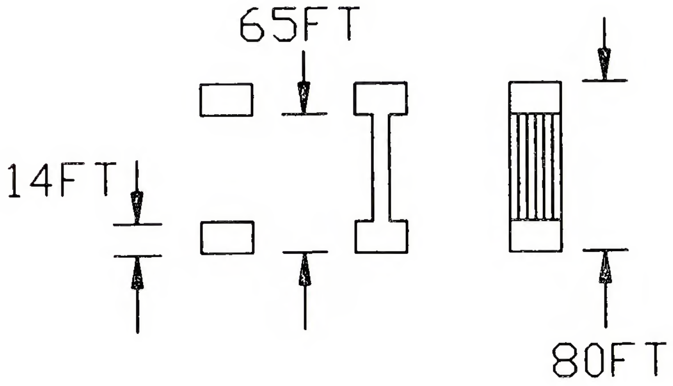
STANDARD LAMEFOOT
BOLTING PATTERN
WASTE HEADINGS

DATE: MAY 1994	SCALE:	FIG. NO. 13

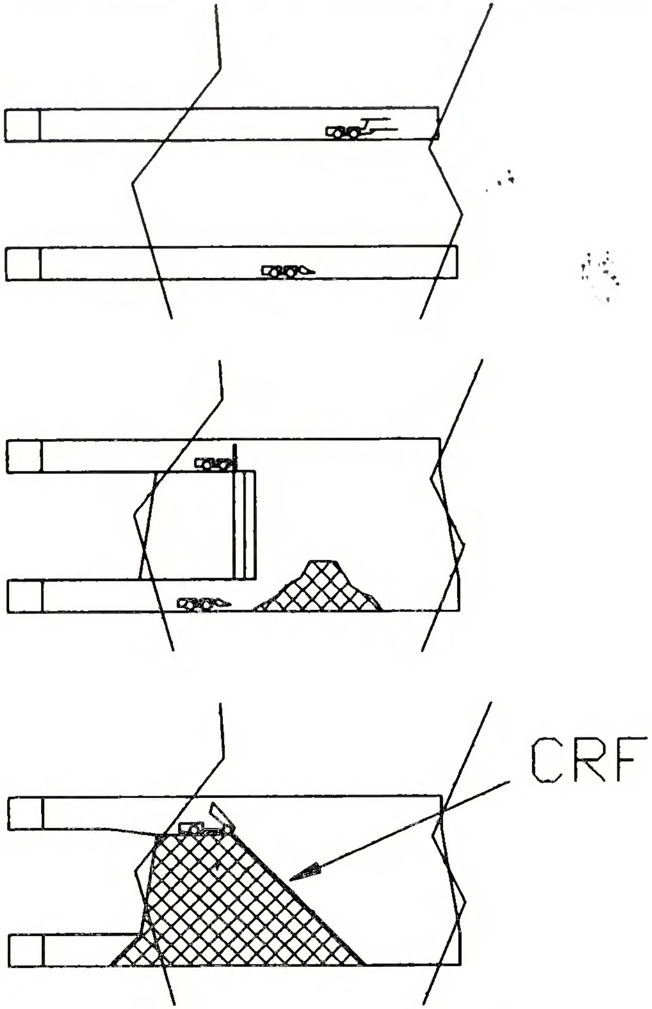
NEILFIG1



PLAN VIEW



LONG SECTION



CROSS SECTION

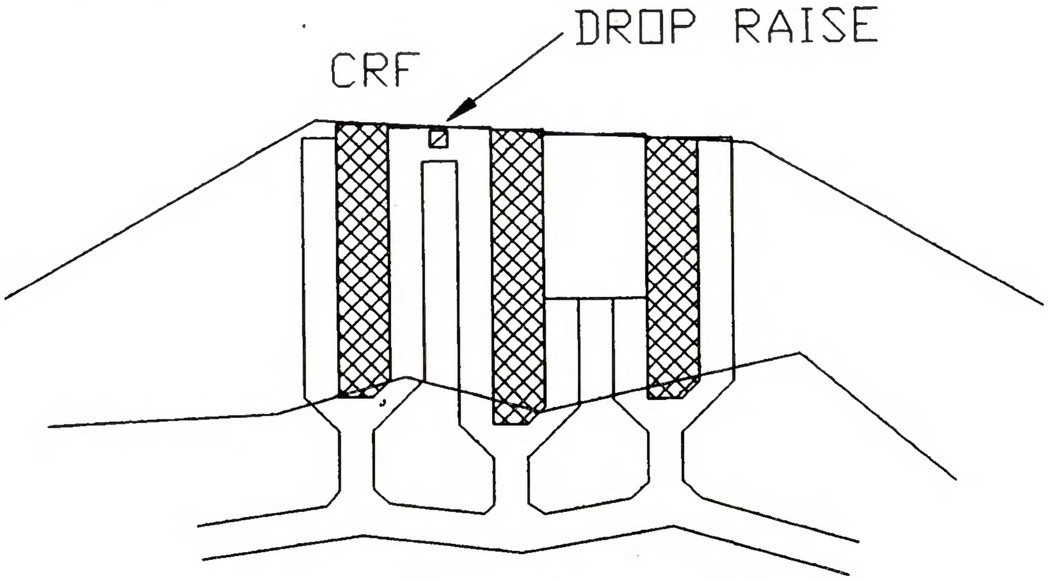
ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

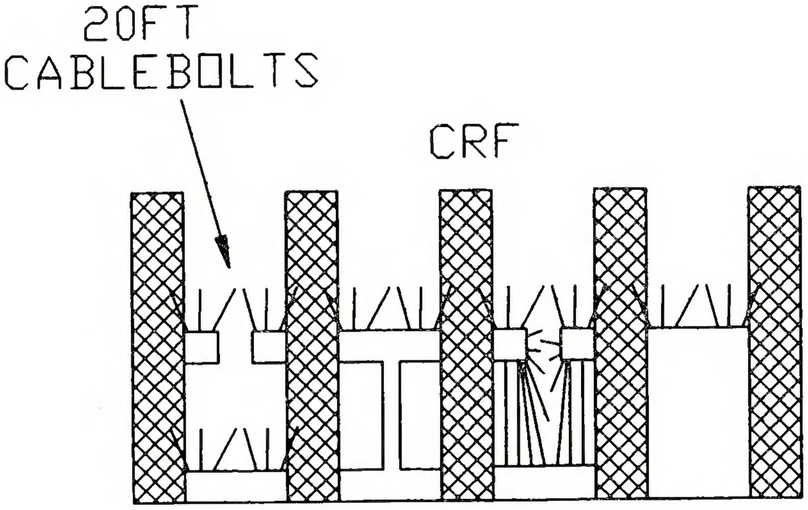
LAMEFOOT PROJECT
TRANSVERSE STOPING
PRIMARY STOPING

DATE: MAY 1994	SCALE: 1" = 90'	FIG. NO. 14

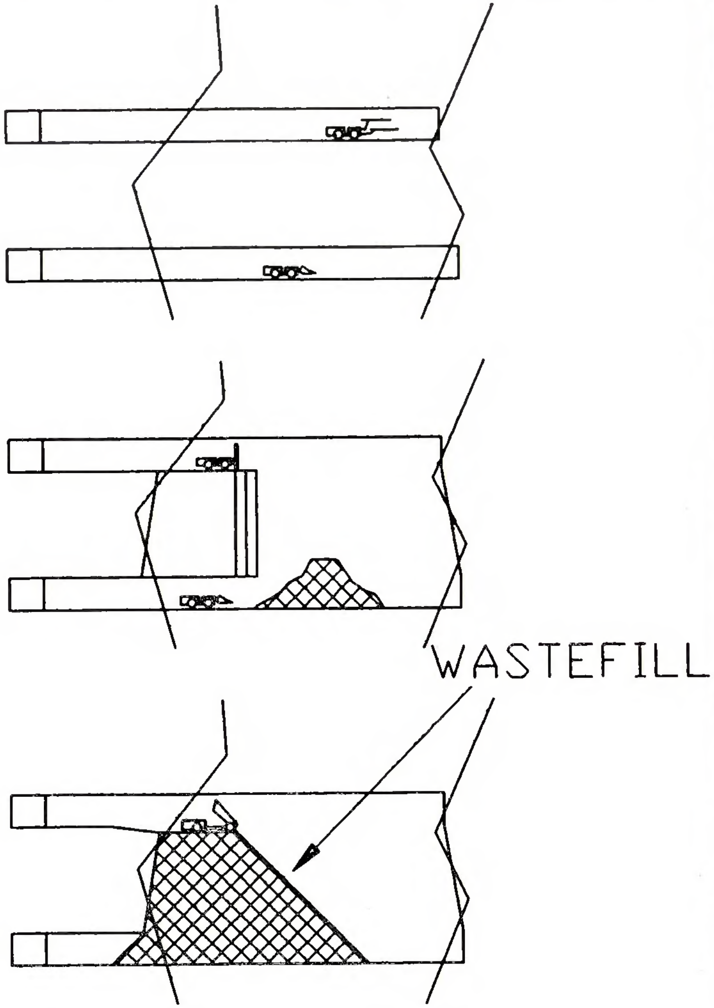
NEILFIG3



PLAN VIEW



LONG SECTION



CROSS SECTION

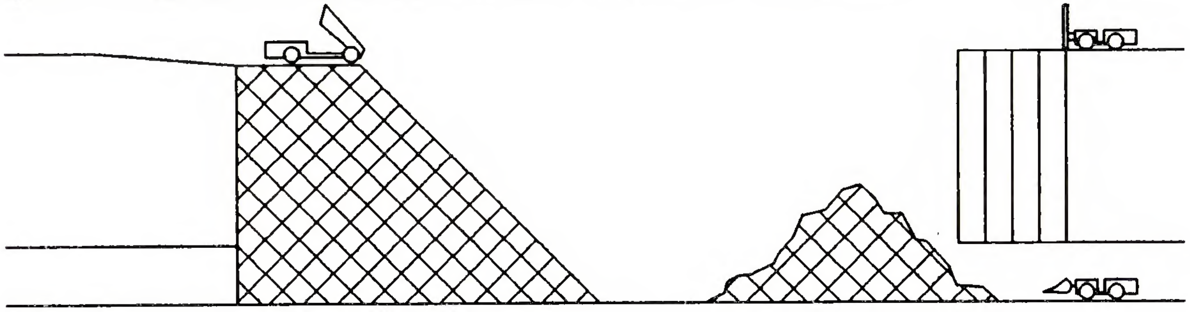
ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

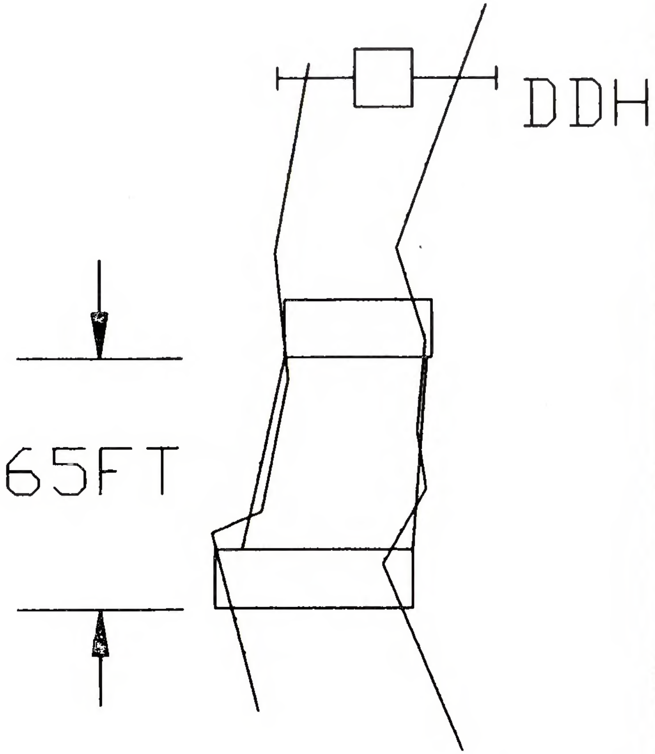
LAMEFOOT PROJECT
TRANVERSE STOPING
SECONDARY STOPING

DATE: MAY 1994	SCALE: 1" = 90'	FIG. NO. 15

NEILFIG4



LONG SECTION



CROSS SECTION

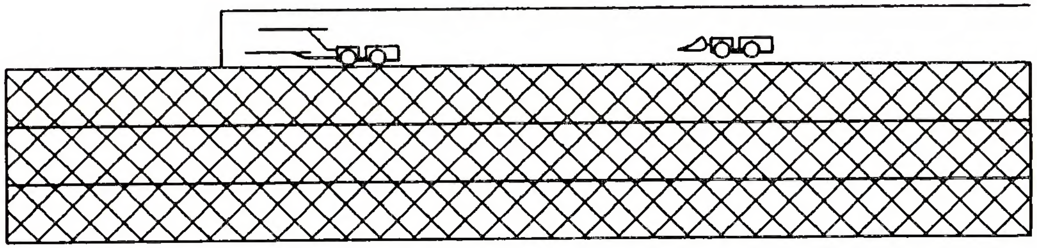
ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

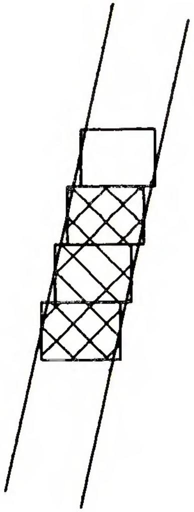
LAMEFOOT PROJECT LONGITUDINAL STOPING

DATE: MAY 1994	SCALE: 1" = 50'	FIG. NO. 16

NEILFIG2



LONG SECTION



CROSS SECTION

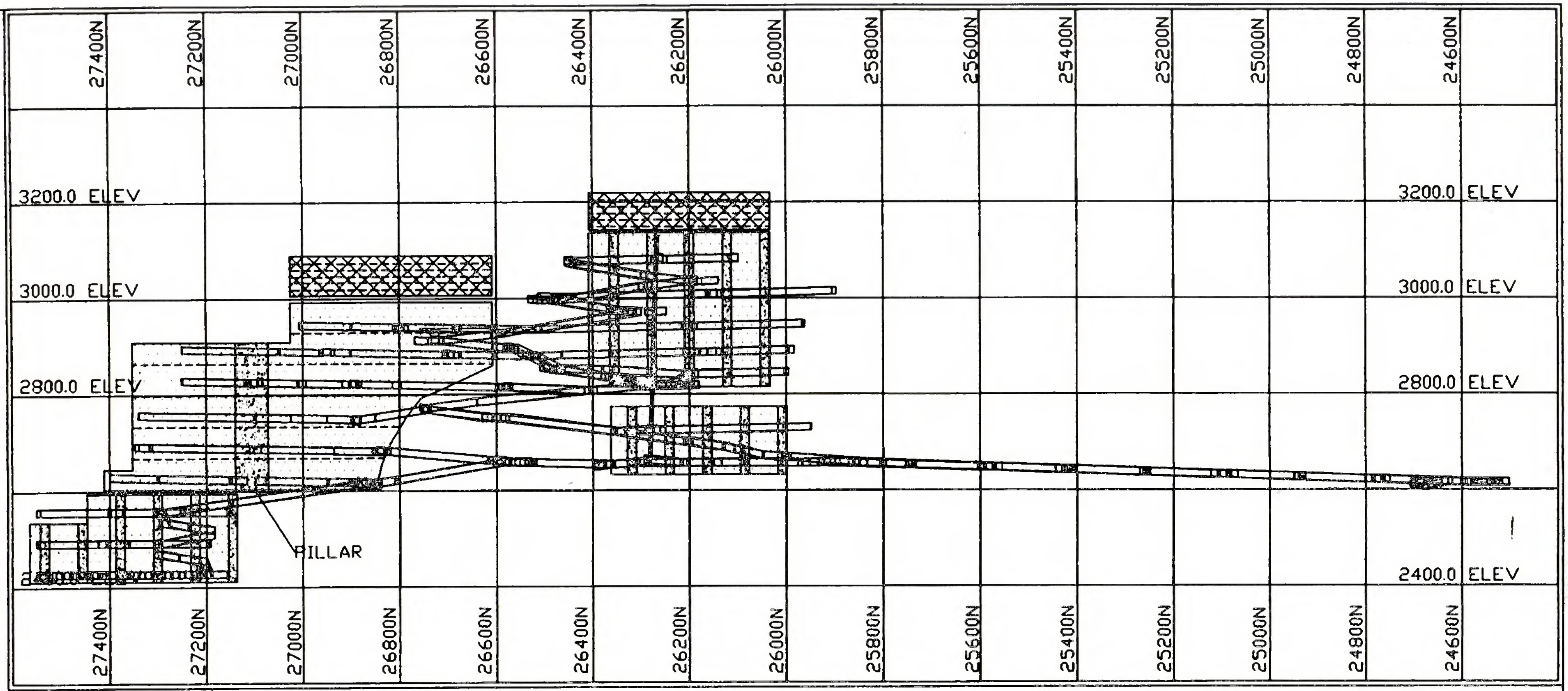
ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

LAMEFOOT PROJECT
DRIFT AND WASTEFILL

DATE: APRIL 1993	SCALE: 1" = 50'	FIG. NO. 17

NEILFIG5



▨ CEMENTED ROCK FILL

▨ TRANSVERSE STOPING

▨ WASTE FILL

▨ LONGITUDINAL STOPING

▨ DRIFT AND WASTE FILL

FILE: 2-12-2

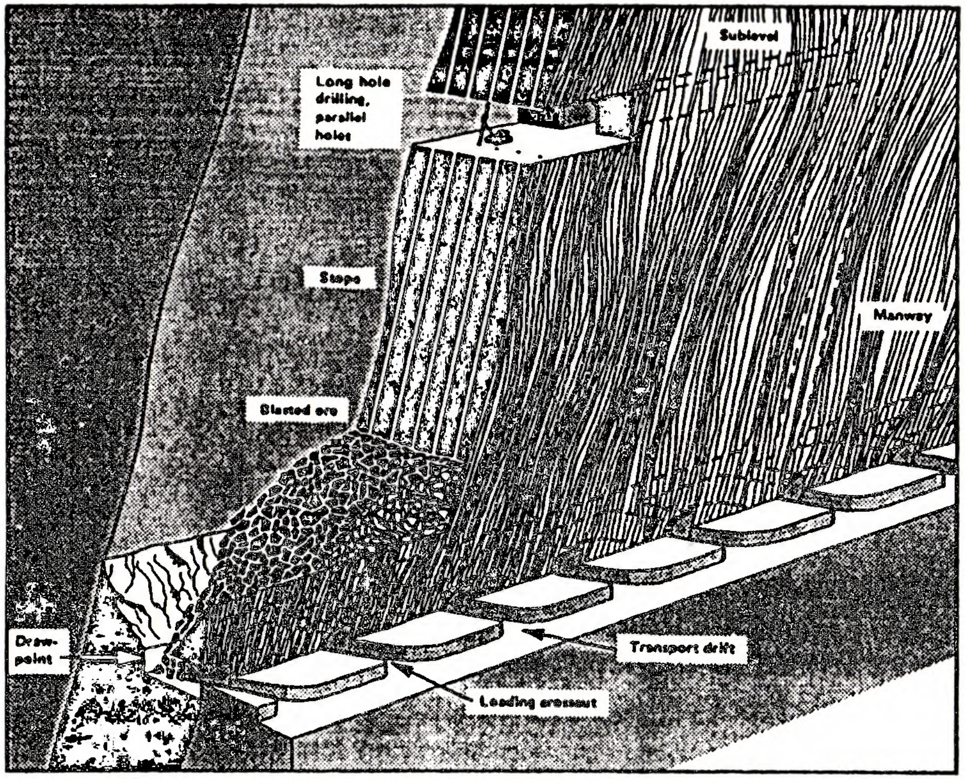
ECHO BAY LAMEFOOT PROJECT

Source: Echo Bay 1993

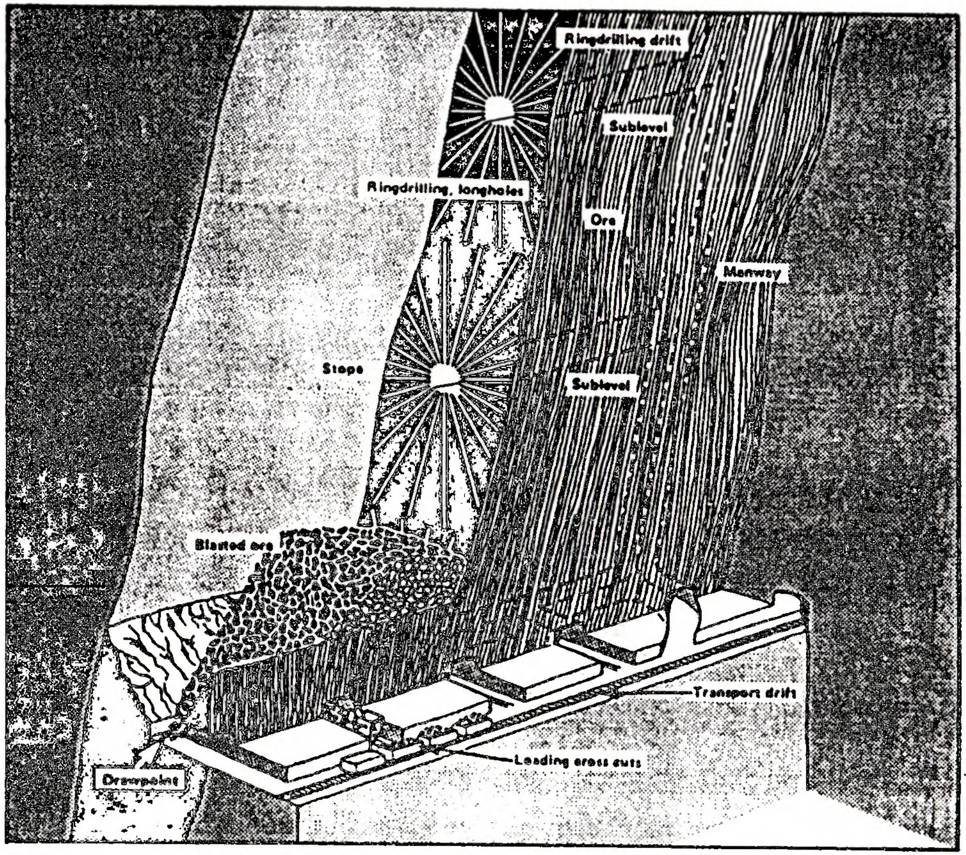
ECHO BAY MINERALS Co.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

STOPING AND BACKFILLING METHODS
LAMEFOOT PROJECT
LONG SECTION LOOKING EAST

DATE: MAY 1994	SCALE	FIG. NO. 18



Sublevel stoping with long-hole drilling and blasting.



Sublevel stoping with ring drilling as the primary means of breaking ore.

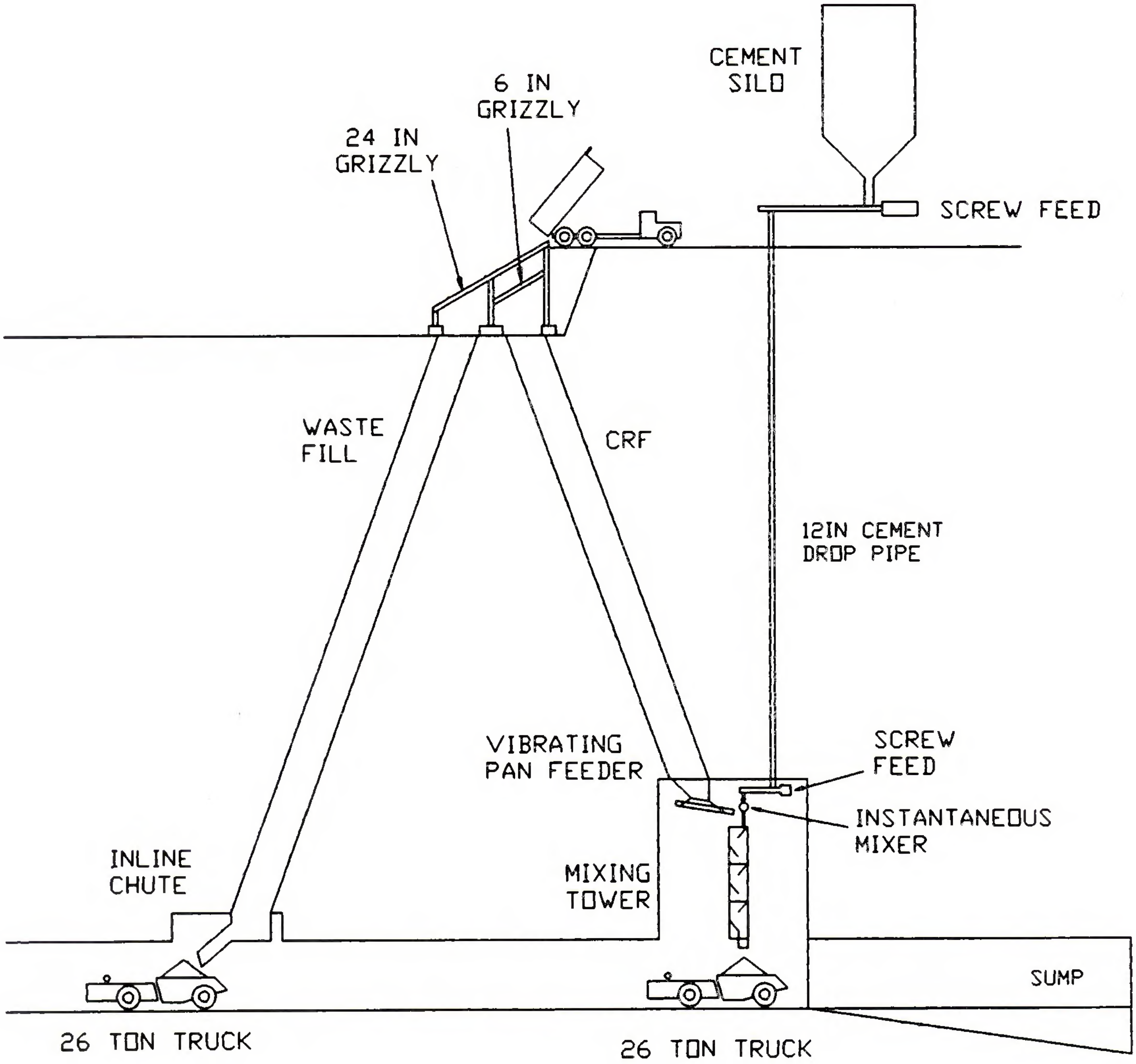
Source: Underground Mining Methods Handbook, Society of Mining Engineers, 1982.

CLIENT/PROJECT
ECHO BAY MINERALS Co.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

ECHO BAY MINERALS, INC.

TITLE
CONCEPTUAL DIAGRAM OF SUBLEVEL STOPING WITH LONG-HOLE AND RING DRILLING AND BLASTING METHODS
SOURCE: ECHO BAY 1993a

DRAWN	CHECKED J.C.	REVIEWED L.E.C.	DATE APRIL 1993	SCALE N/A	FILE NO.	JOB NO. 933-2614	DWG NO./REV.NO.	FIGURE 19



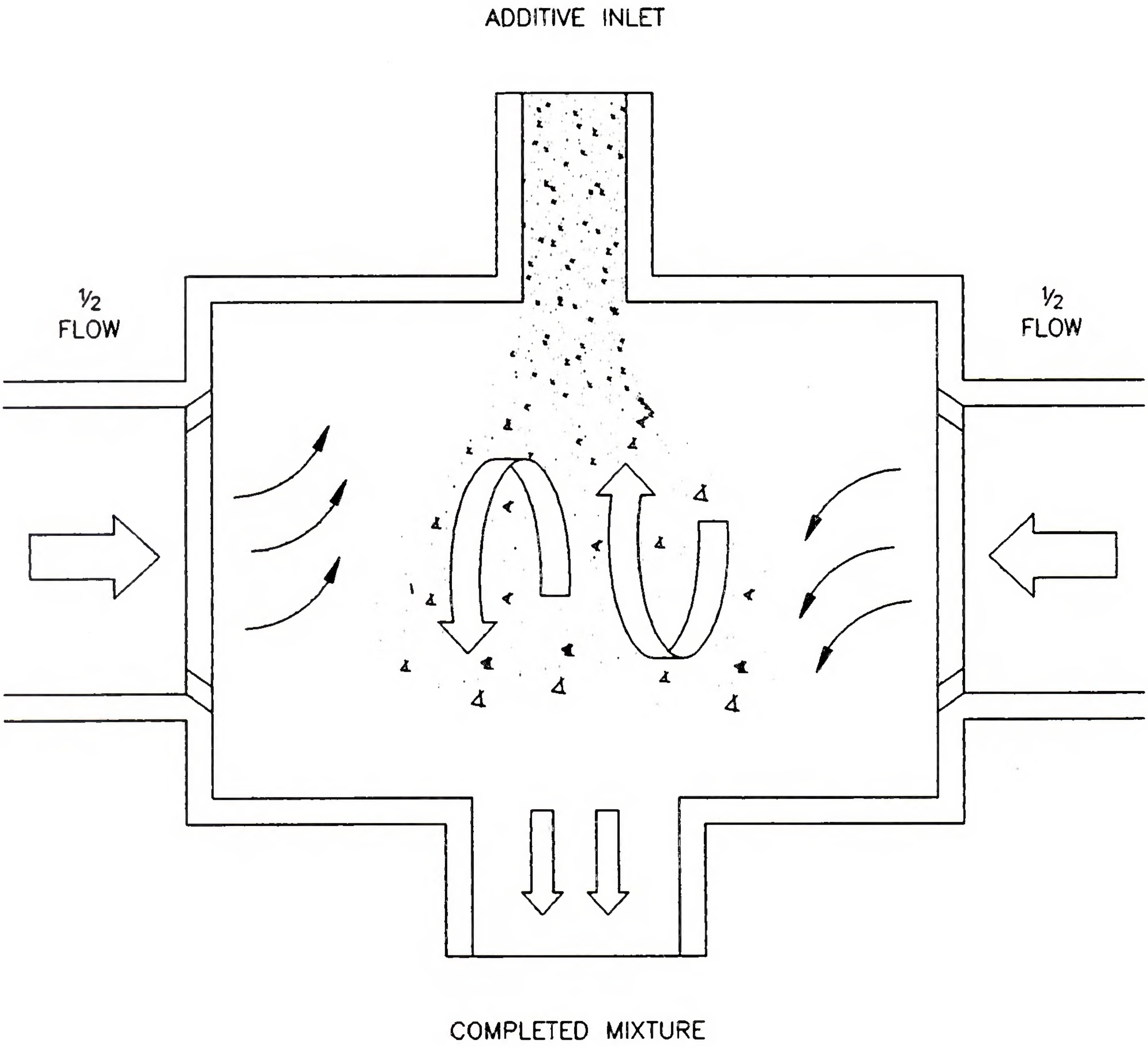
ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

LAMEFOOT PROJECT CEMENTED ROCKFILL PLANT CONCEPTUAL LAYOUT

DATE: MAY 1994 | SCALE: | FIG. NO. 20

NEILFIG6



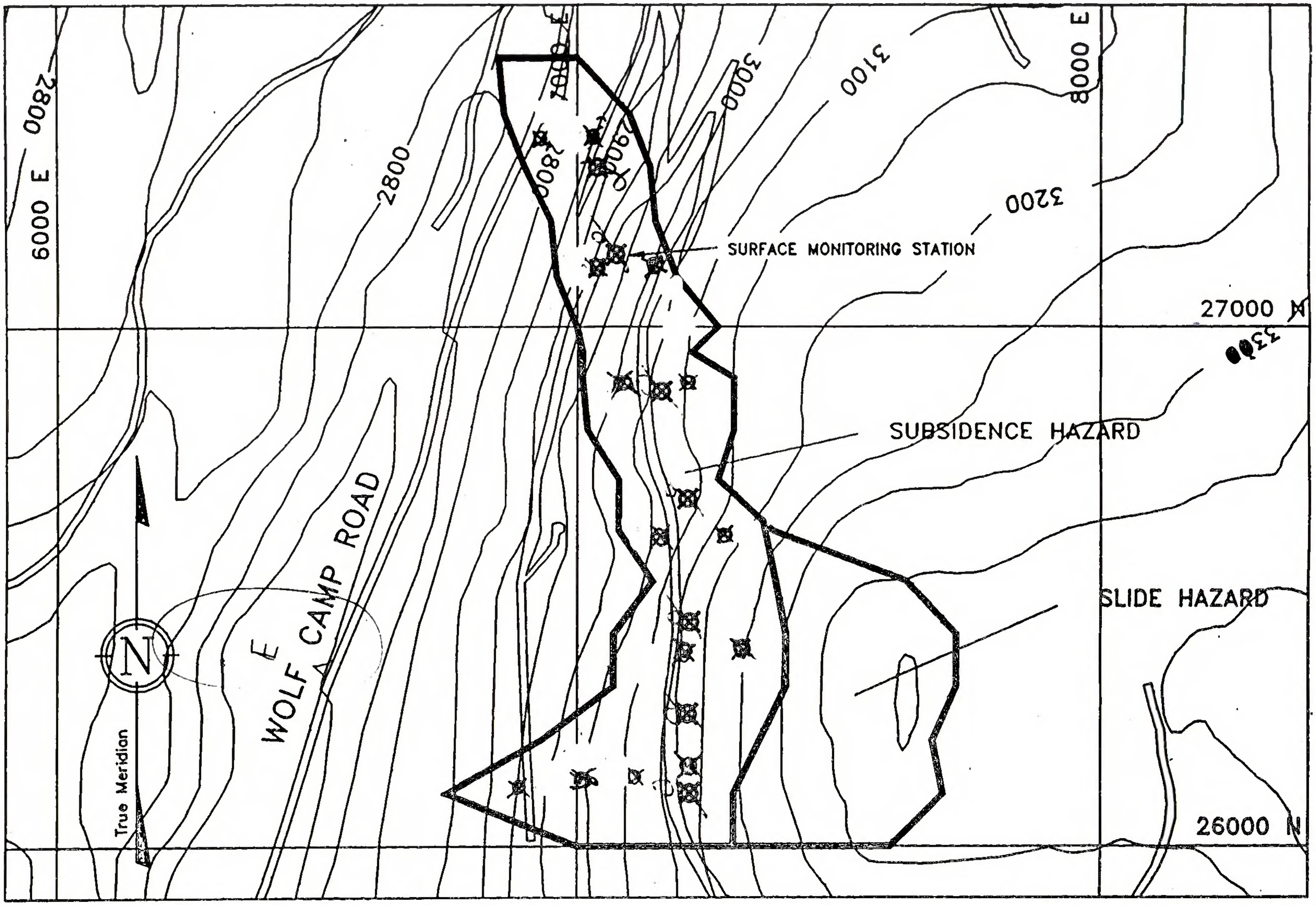
ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

GROUT MIX PLANT

DATE: MAY 1994	SCALE:	FIG. NO. 21

GRMIX



SUBPLAN

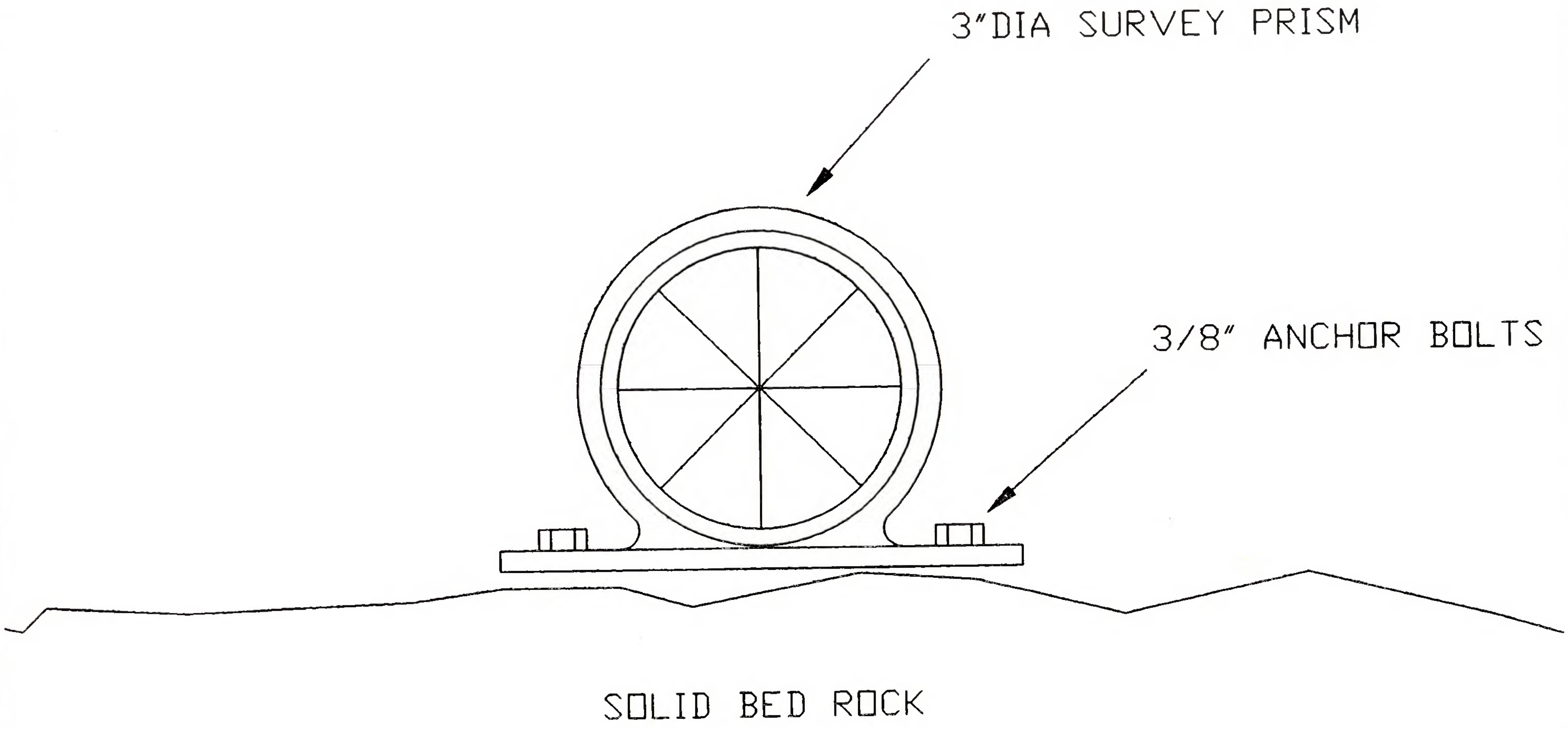
ECHO BAY LAMEFOOT PROJECT

SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

SURFACE MONITORING STATION MAP

DATE: MAY 1994	SCALE: 1:	FIG. NO. 22



ECHO BAY LAMEFOOT PROJECT

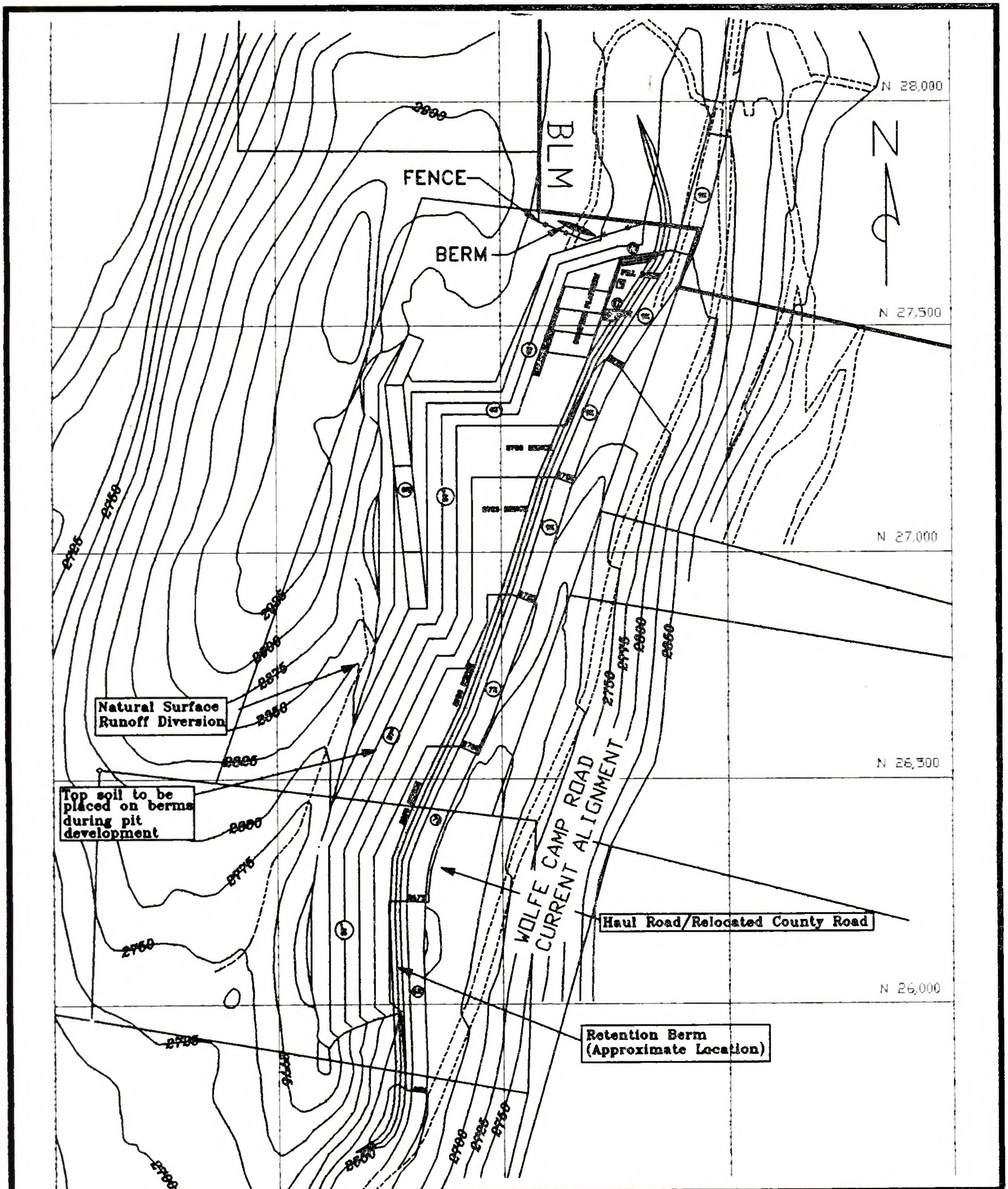
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

SURFACE MONITORING STATION

DATE: MAY 1994	SCALE:	FIG. NO. 23

MONPOINT



N 28,000
N 27,500
N 27,000
N 26,500
N 26,000
N
BLM
FENCE
BERM
2800
2750
2725
2825
2800
2875
2850
2825
2800
2775
2750
2725
2650
2700
2725
2750
2700
2750
2775
2800
2850
Natural Surface Runoff Diversion
Top soil to be placed on berms during pit development
WOLFE CAMP ROAD CURRENT ALIGNMENT
Haul Road/Relocated County Road
Retention Berm (Approximate Location)

ECHO BAY LAMEFOOT PROJECT

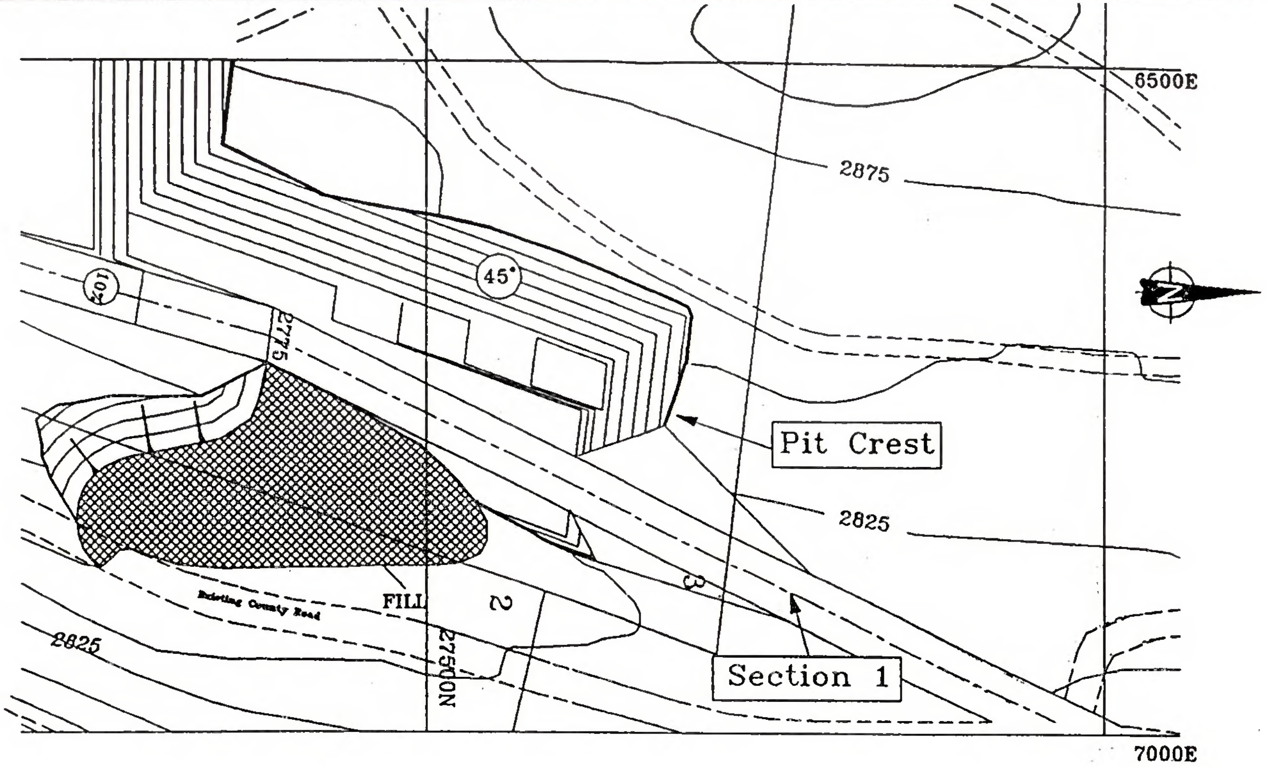
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

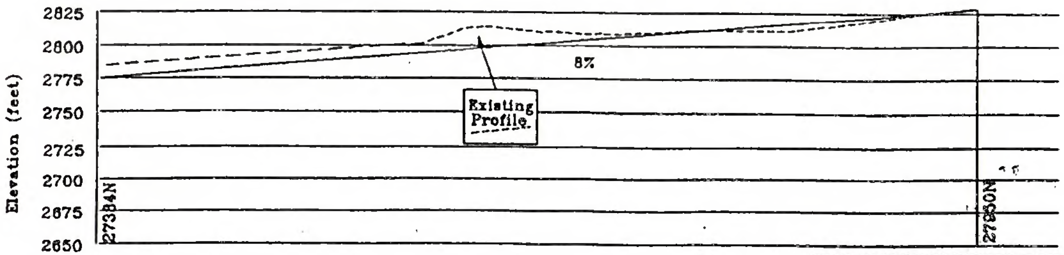
BACKFILL QUARRY AND COUNTY ROAD DIVERSION

DATE: APRIL 1994 SCALE: 1" = 300' FIG. NO. 24

RBQUARRY



Section 1



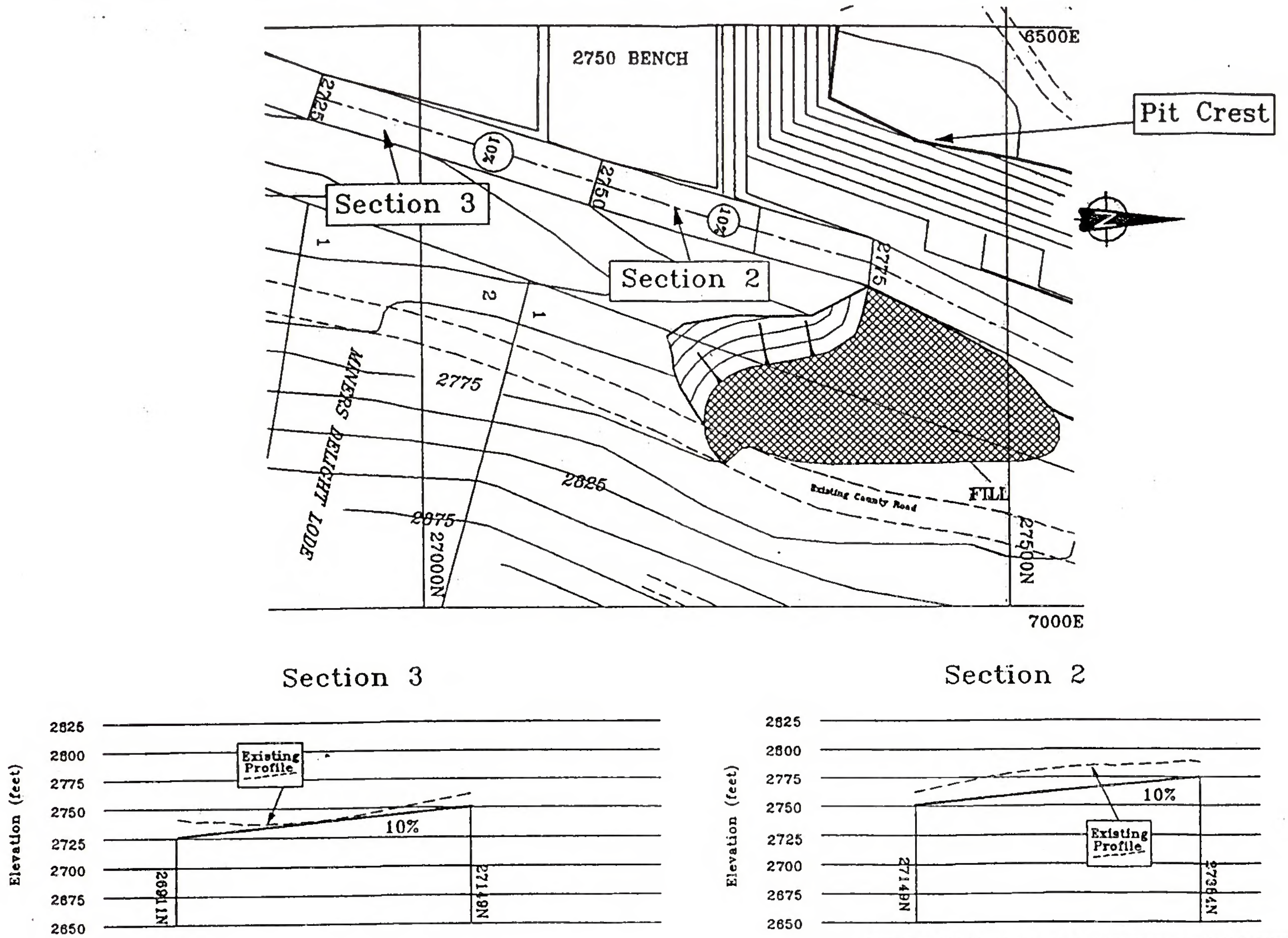
Scale 1"=100'

ECHO BAY LAMEFOOT PROJECT
SOURCE: GOLDER 1992a

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

RELOCATED COUNTY ROAD
SECTION 1
PLAN & PROFILE

DATE: JUNE 1994 SCALE: FIG. NO. 24a



Scale 1"=100'

ECHO BAY LAMEFOOT PROJECT

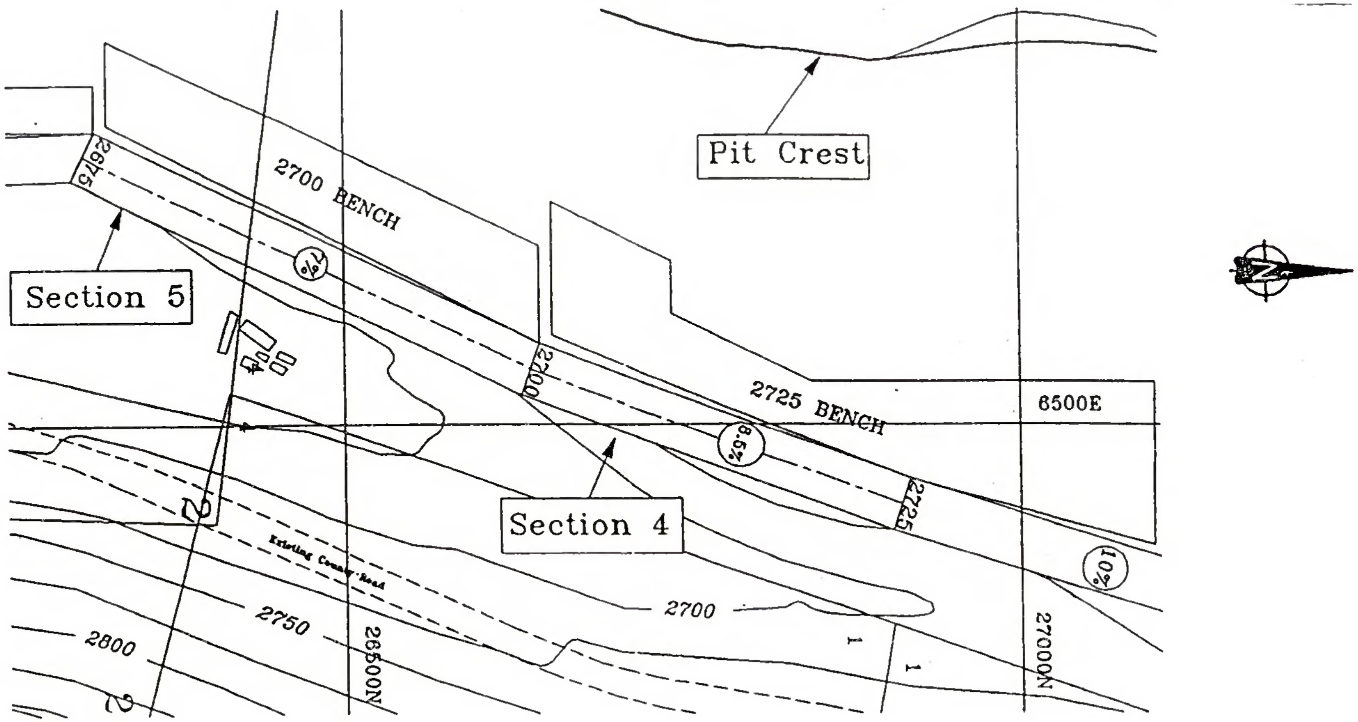
SOURCE: GOLDER 1992a

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

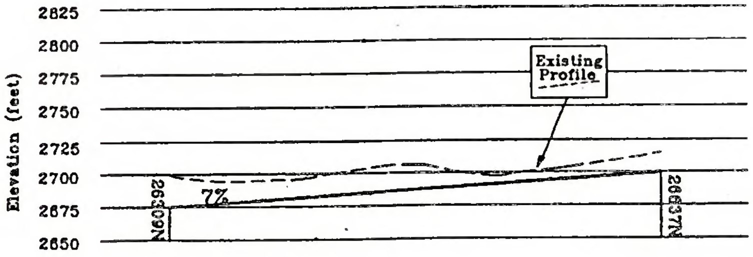
RELOCATED COUNTY ROAD SECTIONS 2 & 3 PLAN & PROFILE

DATE: JUNE 1994 | SCALE: | FIG. NO. 2-6

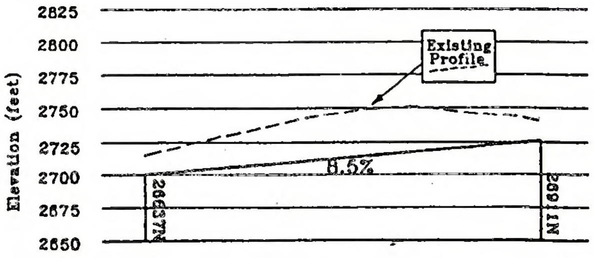
EB1



Section 5



Section 4



Scale 1"=100'

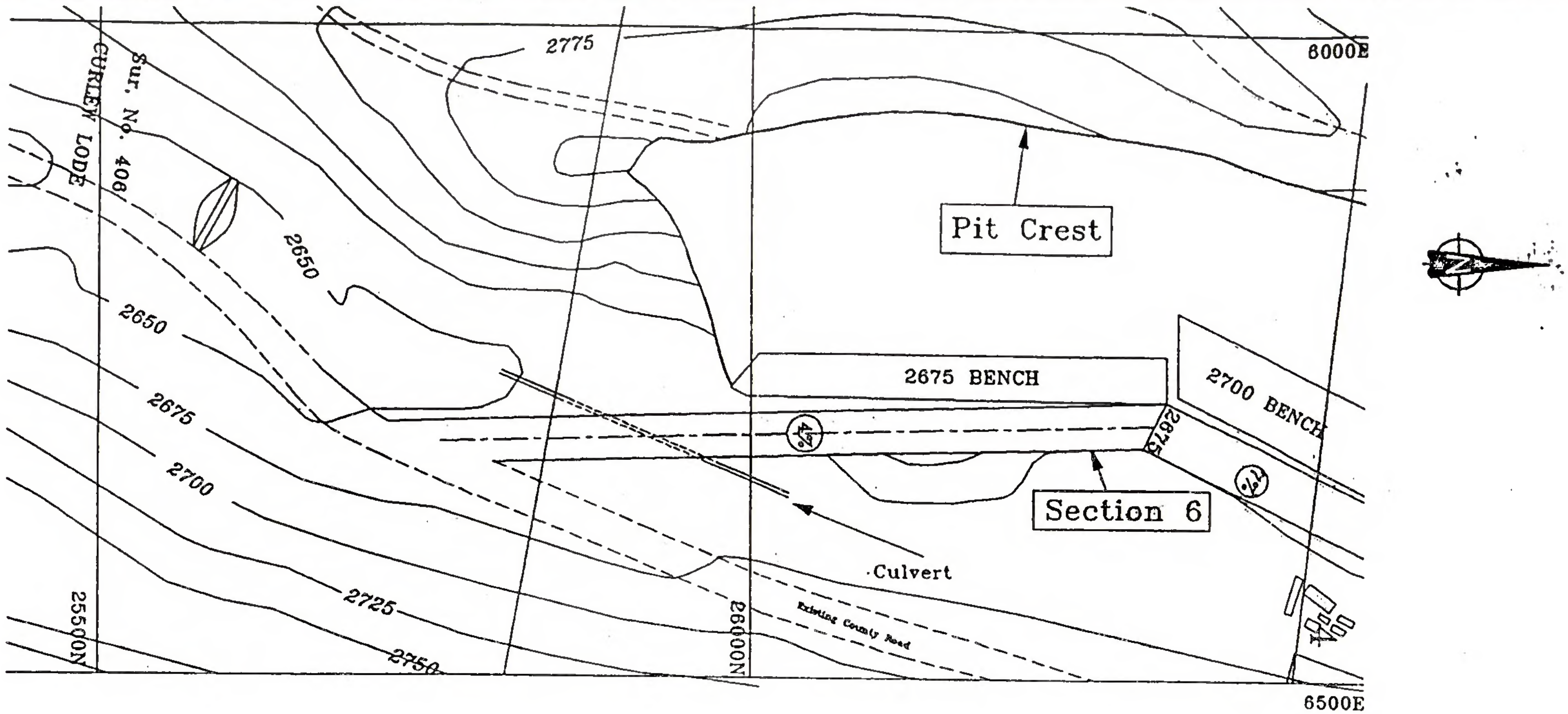
ECHO BAY LAMEFOOT PROJECT

SOURCE: GOLDER 1992a

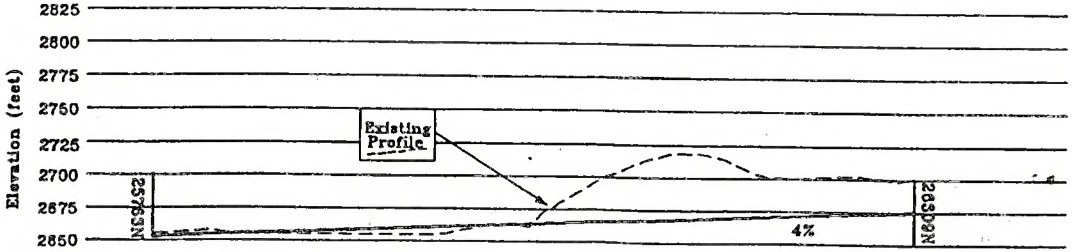
ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

RELOCATED COUNTY ROAD
SECTIONS 4 & 5
PLAN & PROFILE

DATE: JUNE 1994	SCALE:	FIG. NO. 24c



Section 6



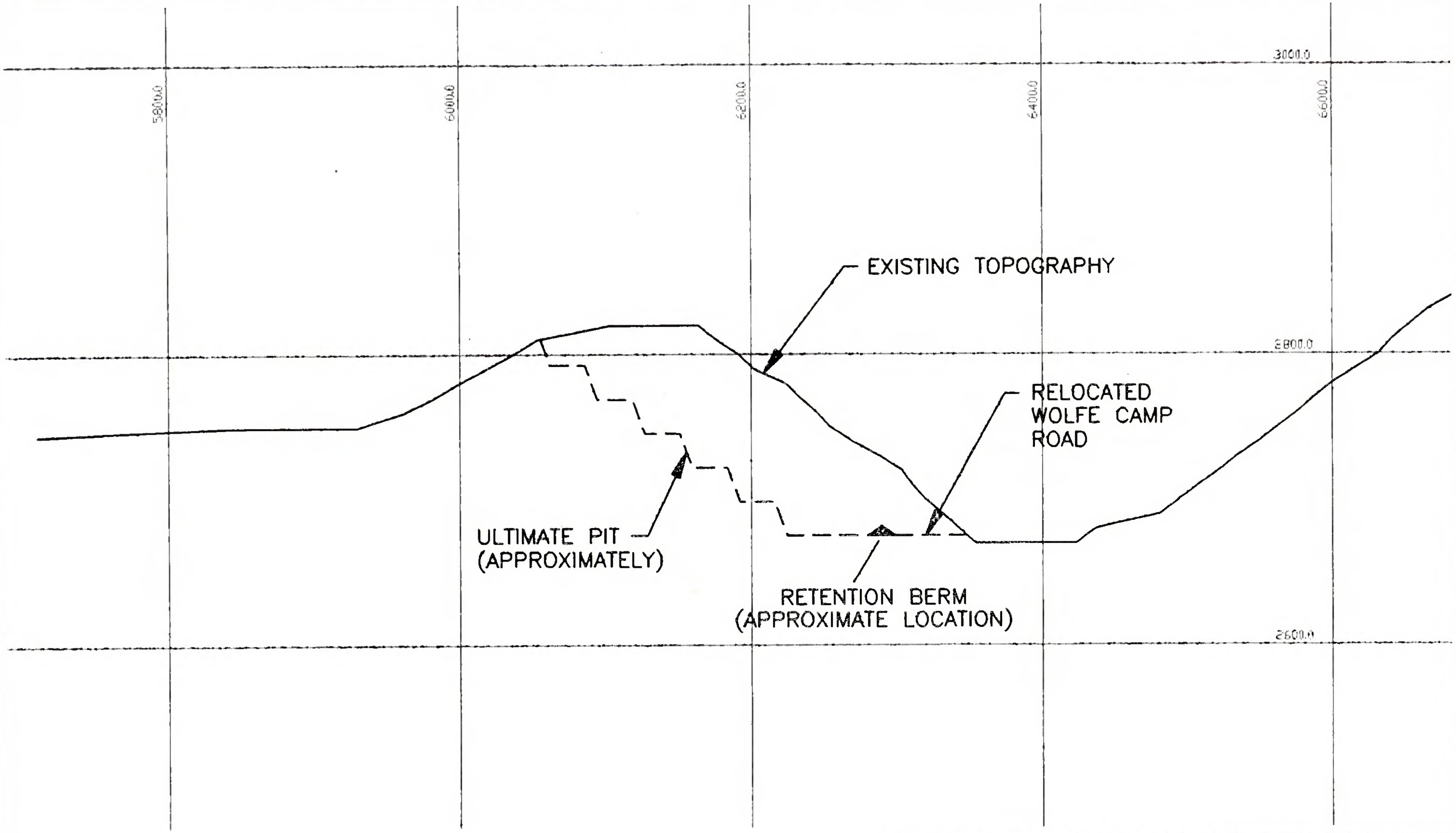
ECHO BAY LAMEFOOT PROJECT

SOURCE: GOLDER 1992a

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

RELOCATED COUNTY ROAD SECTION 6 PLAN & PROFILE

DATE: JUNE 1994	SCALE:	FIG. NO. 24d


5800.0
6000.0
6200.0
6400.0
6600.0
3000.0
2800.0
2600.0
EXISTING TOPOGRAPHY
RELOCATED WOLFE CAMP ROAD
ULTIMATE PIT (APPROXIMATELY)
RETENTION BERM (APPROXIMATE LOCATION)

ECHO BAY LAMEFOOT PROJECT

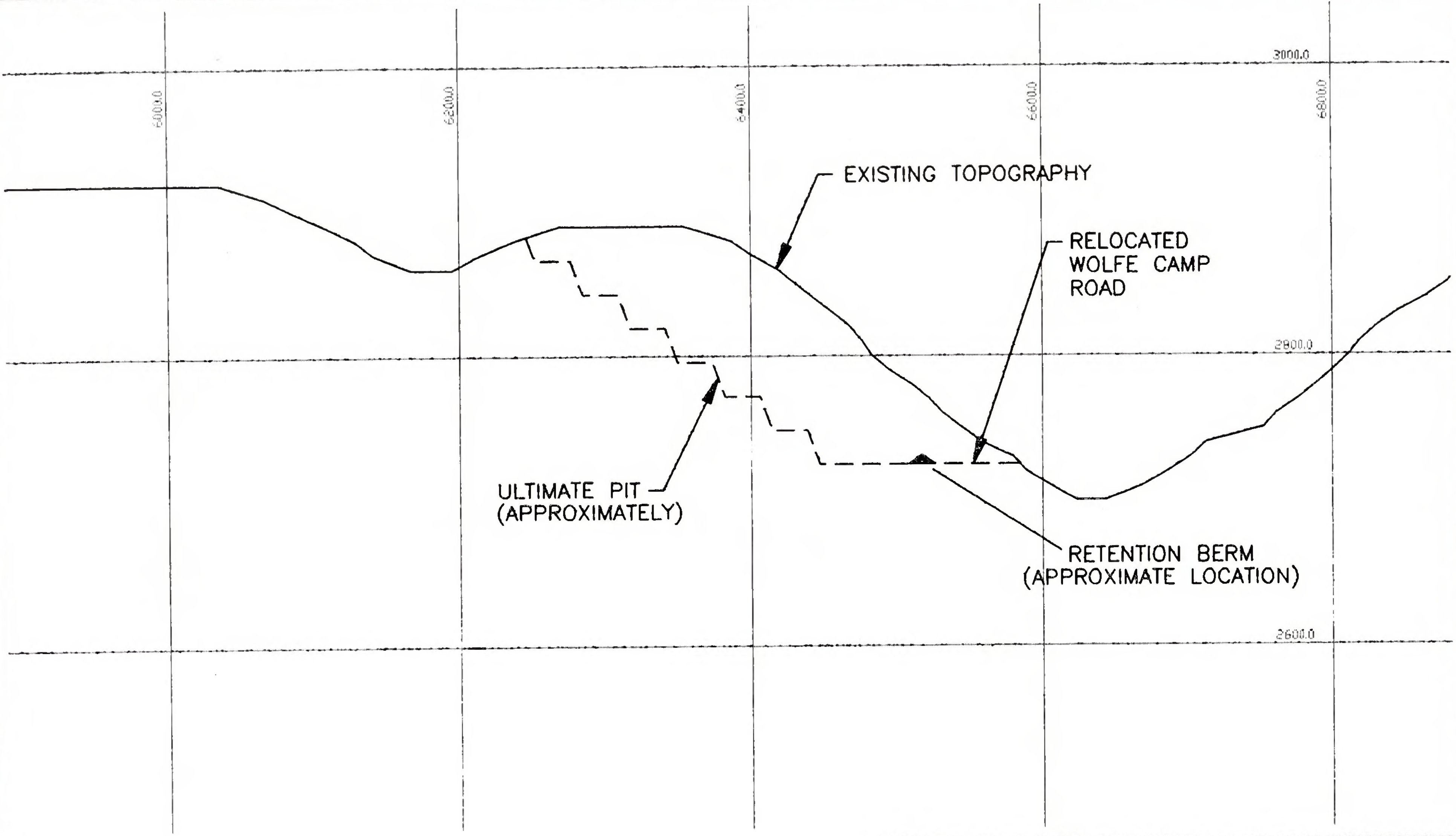
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

BACKFILL QUARRY SECTION 26100N

DATE: APRIL 1994 | SCALE: 1" = 100' | FIG. NO. 25

RBX26100


3000.0
2800.0
2600.0
6000.0
6200.0
6400.0
6600.0
6800.0
EXISTING TOPOGRAPHY
RELOCATED WOLFE CAMP ROAD
ULTIMATE PIT (APPROXIMATELY)
RETENTION BERM (APPROXIMATE LOCATION)

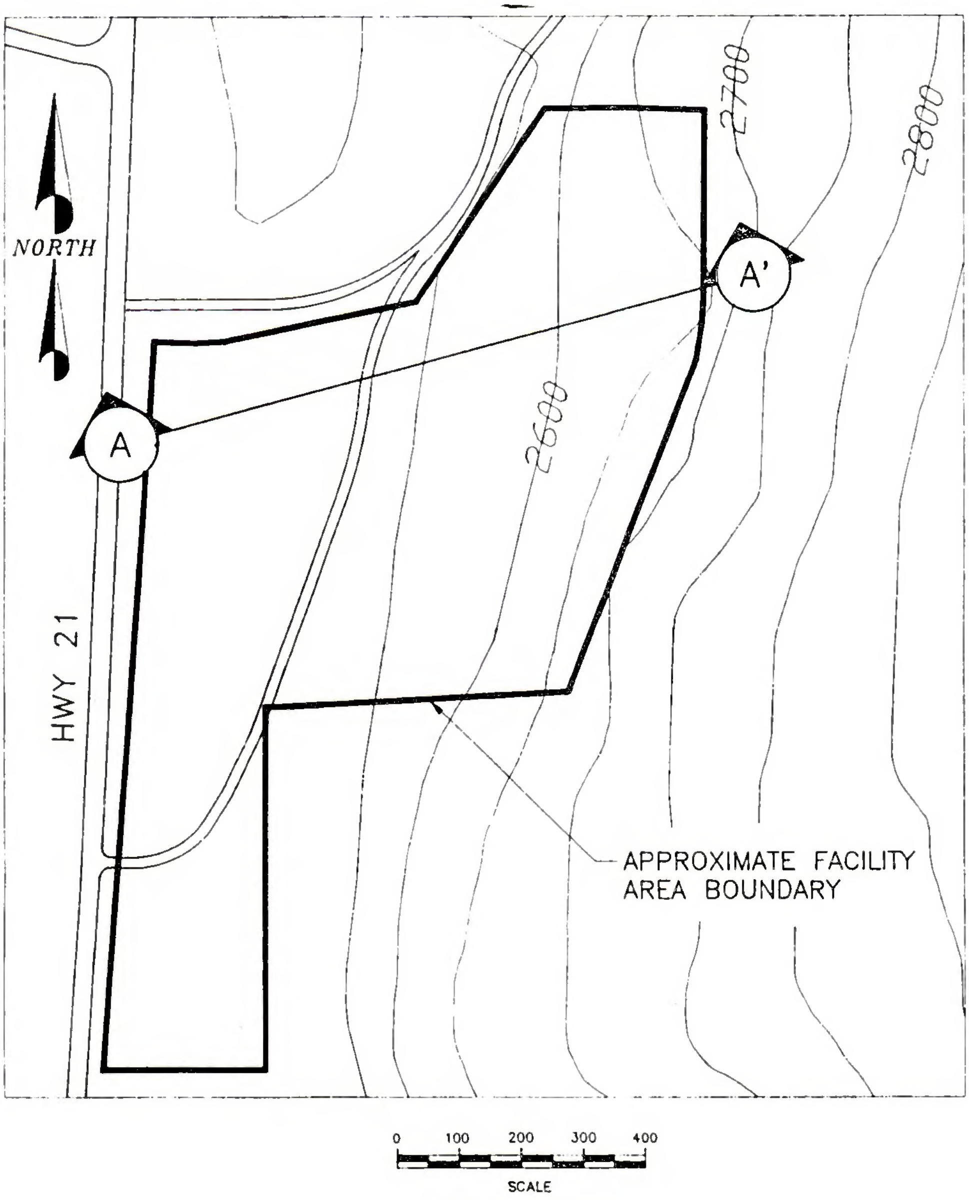
ECHO BAY LAMEFOOT PROJECT

SOURCE: ECHO BAY 1993

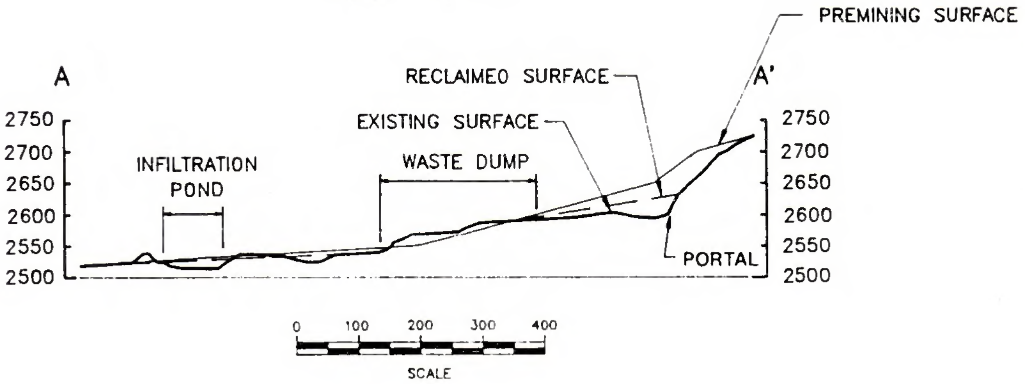
ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

BACKFILL QUARRY
SECTION 26900N

DATE: APRIL 1994	SCALE: 1" = 100'	FIG. NO. 26



SCALE: 1" = 200'



SCALE: 1" = 200'
NO VERTICAL EXAGGERATION

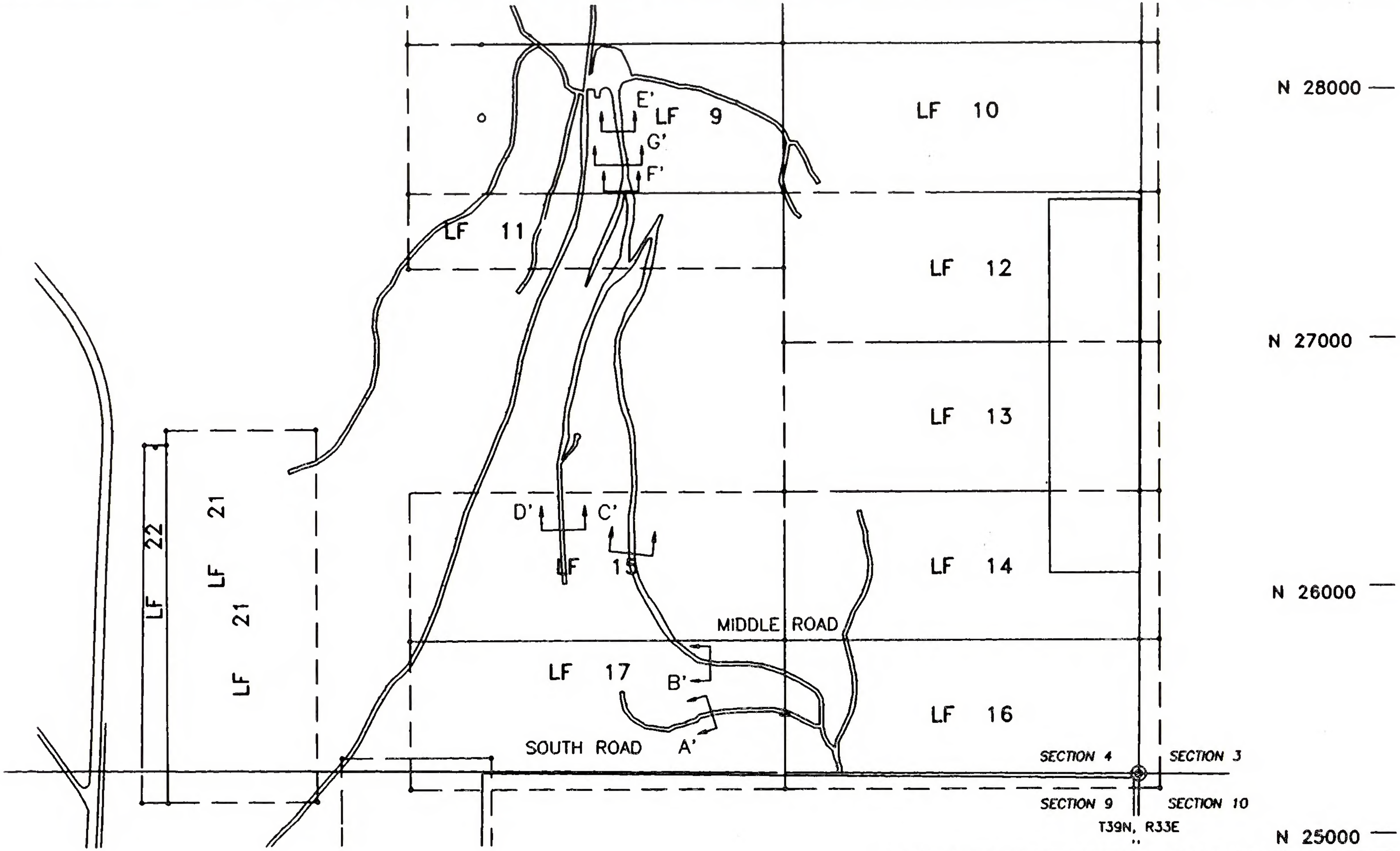
ECHO BAY LAMEFOOT PROJECT

ECHO BAY MINERALS CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

PROPOSED FINAL RECLAIMED TOPOGRAPHY OF THE LAMEFOOT FACILITIES AREA

DATE MAY 1993	SCALE 1"=200'	FIG. NO. 2 7

FILE: B-16



ECHO BAY LAMEFOOT PROJECT

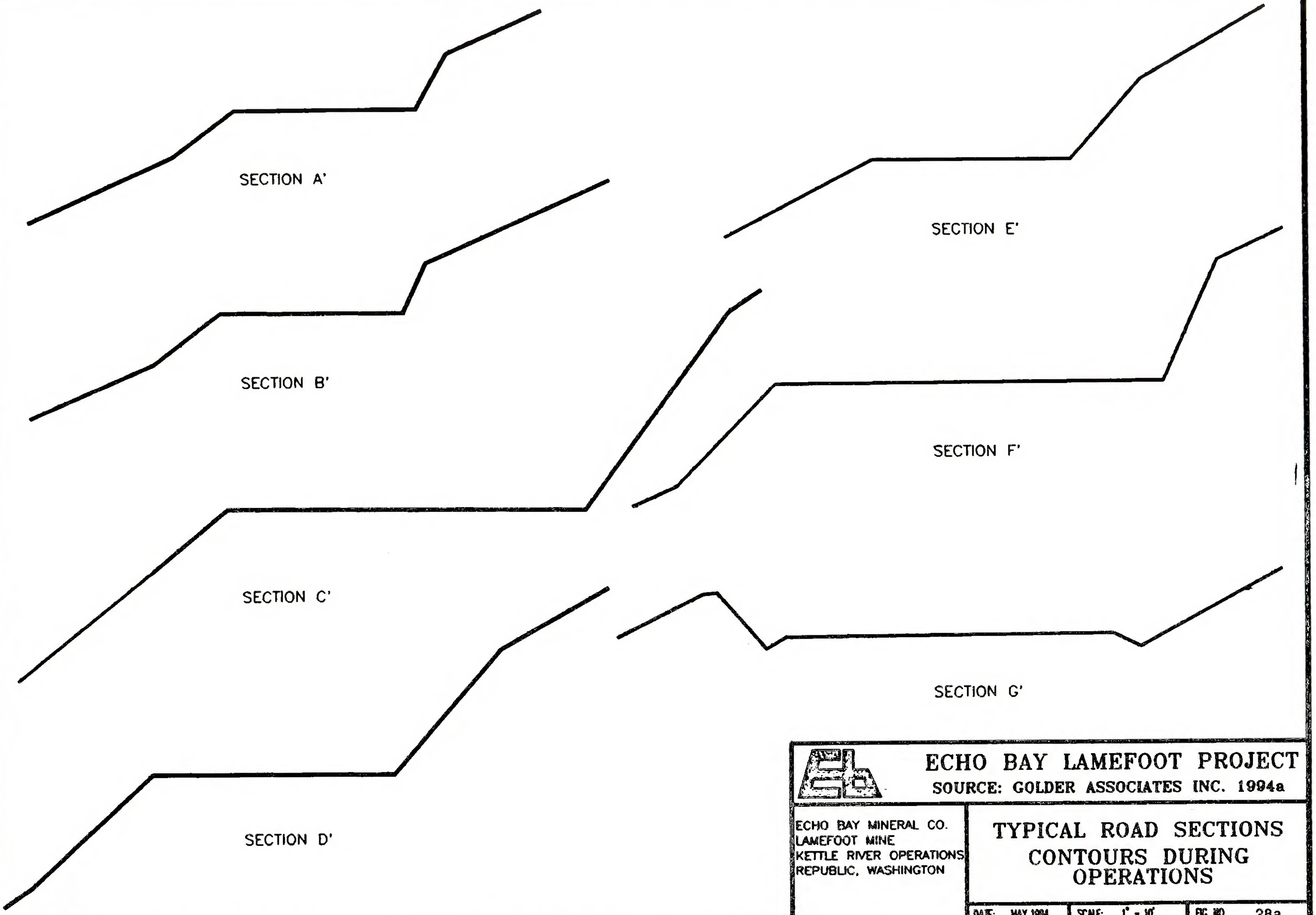
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

LOCATION MAP FOR TYPICAL ROAD SECTIONS

DATE: MAY 1994 | SCALE: 1" = 600' | FIG. NO. 28

SECLOCMP



ECHO BAY LAMEFOOT PROJECT

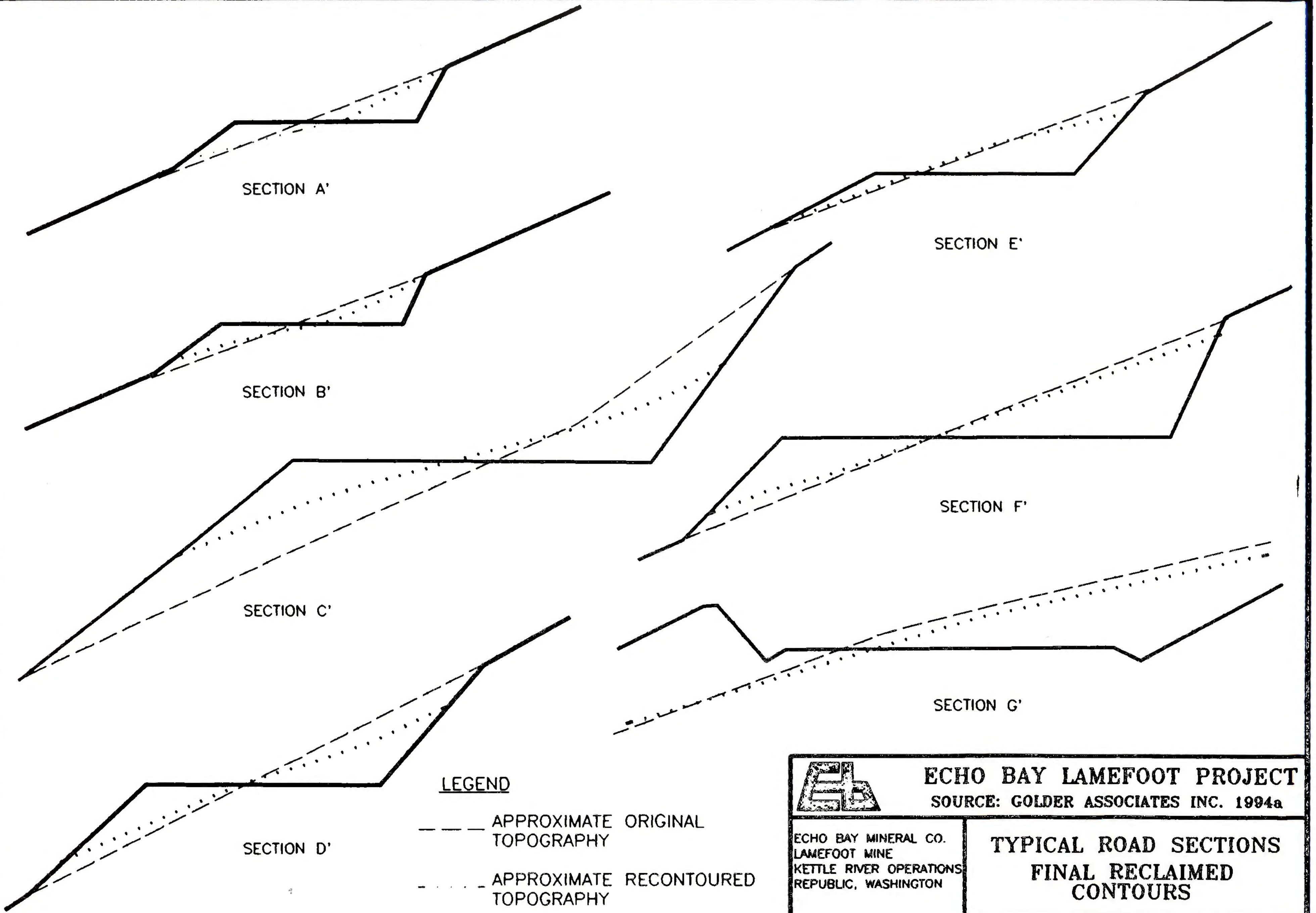
SOURCE: GOLDER ASSOCIATES INC. 1994a

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

TYPICAL ROAD SECTIONS CONTOURS DURING OPERATIONS

DATE:	MAY 1994	SCALE:	1" = 10'	FIG. NO.	28a

LAMTRS



LEGEND

– – – – APPROXIMATE ORIGINAL TOPOGRAPHY

– · · · – APPROXIMATE RECONTOURED TOPOGRAPHY

ECHO BAY LAMEFOOT PROJECT

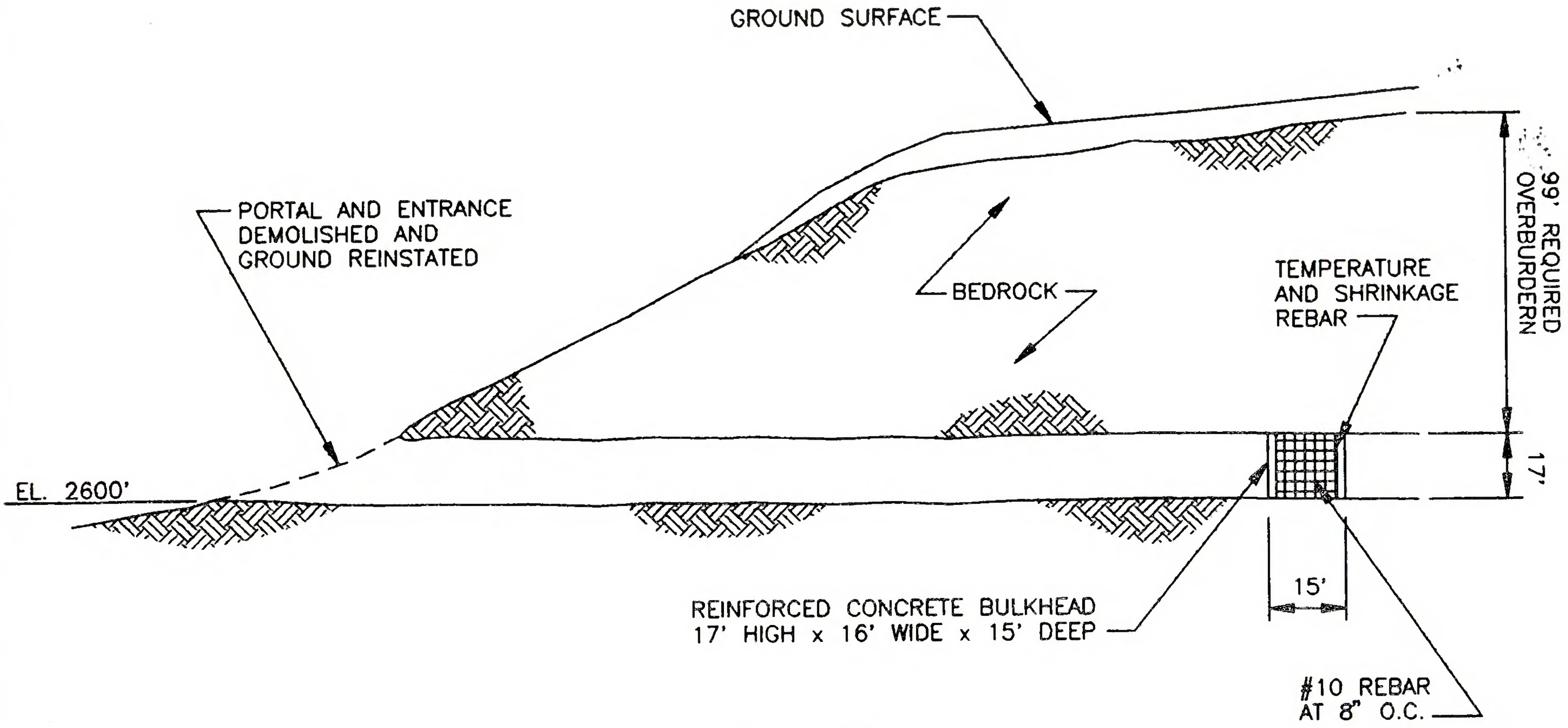
SOURCE: GOLDER ASSOCIATES INC. 1994a

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

TYPICAL ROAD SECTIONS FINAL RECLAIMED CONTOURS

DATE: MAY 1994	SCALE: 1" = 10'	FIG. NO. 28b

LAMTRS1


GROUND SURFACE
PORTAL AND ENTRANCE DEMOLISHED AND GROUND REINSTATED
BEDROCK
TEMPERATURE AND SHRINKAGE REBAR
99' REQUIRED OVERBURDERN
17'
EL. 2600'
15'
REINFORCED CONCRETE BULKHEAD 17' HIGH x 16' WIDE x 15' DEEP
#10 REBAR AT 8" O.C.

SIDE VIEW

SCALE: 1" = 50'

NOTES:

1. MAXIMUM HEAD (FT.) 530
2. LINE OF SIGHT DISTANCES FOR ESTIMATING MAXIMUM EARTHQUAKE LOADING APPROXIMATE, BUT CONSERVATIVE. ALL CONCRETE TO BE 3000 PSI COMPRESSIVE STRENGTH.

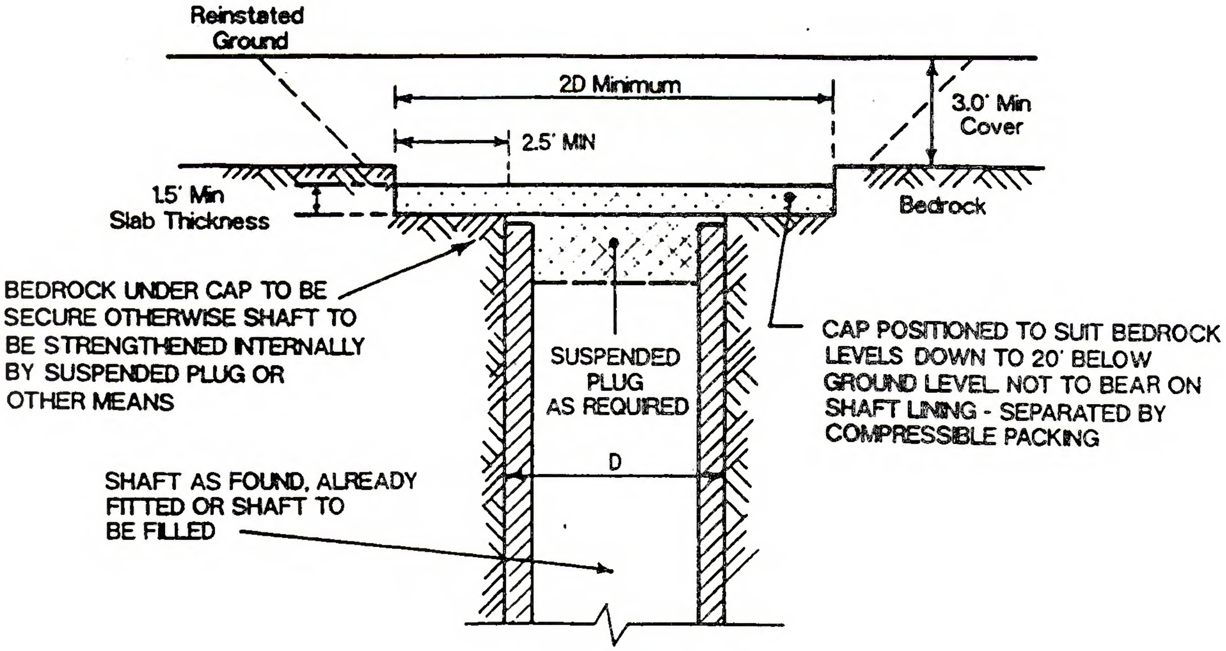
ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1994

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

CONCEPTUAL DIAGRAM OF PORTAL #1 ADIT CLOSURE

DATE: MAY 1994 | SCALE: 1" = 50' | FIG. NO. 29

ECHOBAY2


Reinstated Ground
2D Minimum
2.5' MIN
3.0' Min Cover
1.5' Min Slab Thickness
Bedrock
BEDROCK UNDER CAP TO BE SECURE OTHERWISE SHAFT TO BE STRENGTHENED INTERNALLY BY SUSPENDED PLUG OR OTHER MEANS
SUSPENDED PLUG AS REQUIRED
CAP POSITIONED TO SUIT BEDROCK LEVELS DOWN TO 20' BELOW GROUND LEVEL. NOT TO BEAR ON SHAFT LINING - SEPARATED BY COMPRESSIBLE PACKING
D
SHAFT AS FOUND, ALREADY FITTED OR SHAFT TO BE FILLED

NOTES:
- CAP DESIGNED FOR OVERBURDEN PLUS SUPERIMPOSED LOAD OF 700 Lbs./Ft.^{2}
- REINFORCED CONCRETE SLAB STRENGTH 4350 psi. CONCRETE MINIMUM THICKNESS 1.5'
- REINFORCING BARS TO BE NOT LESS THAN 1 5/8" DIAMETER AND TO HAVE NOT LESS THAN 4 INCHES COVER.
- CAP TO BE MARKED WITH SHAFT REFERENCE NUMBER, DIAMETER AND DEPTH, AND POSITION OF SHAFT CENTER.

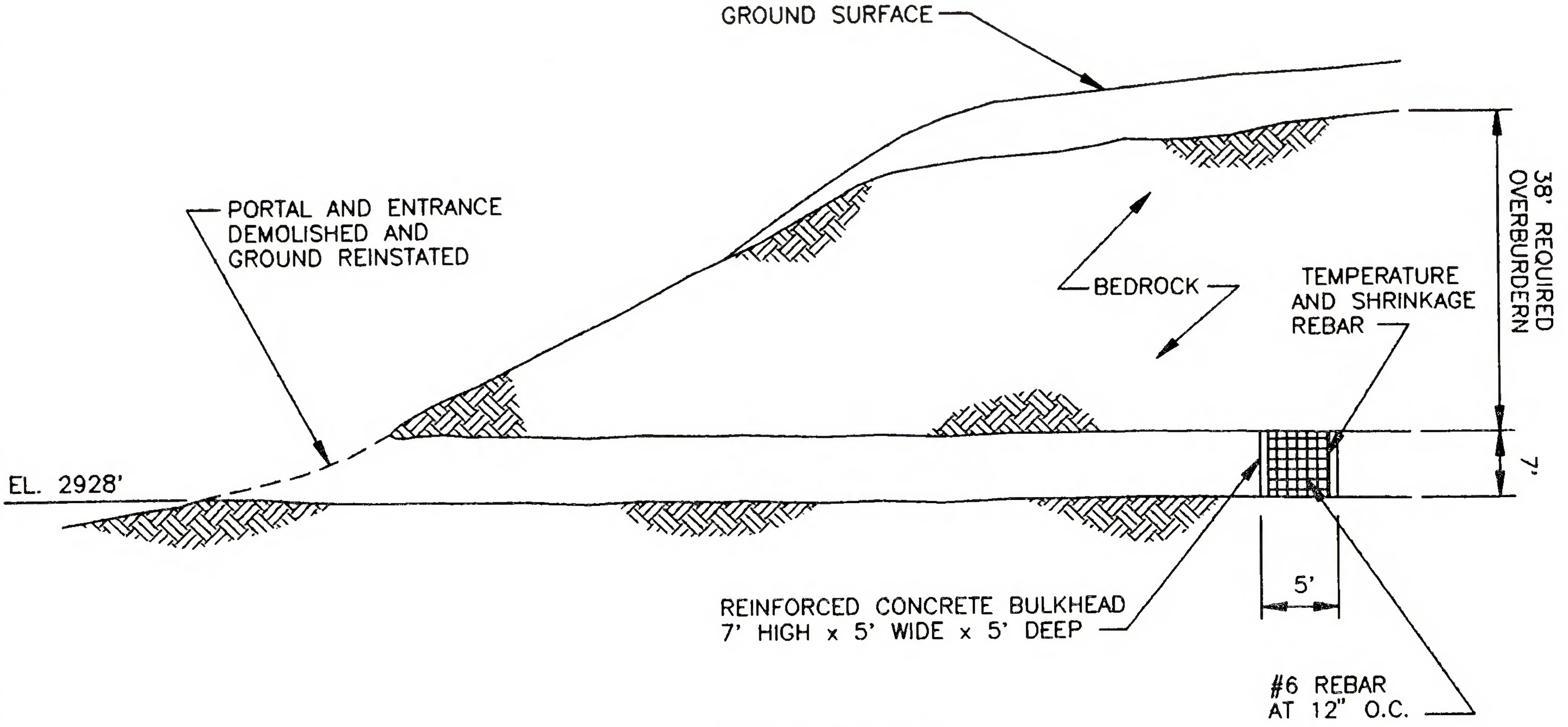
SOURCE: BLM SOLID MINERALS RECLAMATION HANDBOOK, 1992, Figure IX-8

ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

CONCEPTUAL DIAGRAM OF PERMANENT SHAFT CLOSURE

DATE: APRIL 1993 | SCALE: 1" = 200' | FIG. NO. 30


GROUND SURFACE
PORTAL AND ENTRANCE DEMOLISHED AND GROUND REINSTATED
BEDROCK
TEMPERATURE AND SHRINKAGE REBAR
38' REQUIRED OVERBURDERN
7'
EL. 2928'
5'
REINFORCED CONCRETE BULKHEAD 7' HIGH x 5' WIDE x 5' DEEP
#6 REBAR AT 12" O.C.

SIDE VIEW

NOT TO SCALE

NOTES:

1. MAXIMUM HEAD (FT.) 202
2. LINE OF SIGHT DISTANCES FOR ESTIMATING MAXIMUM EARTHQUAKE LOADING APPROXIMATE, BUT CONSERVATIVE. ALL CONCRETE TO BE 3000 PSI COMPRESSIVE STRENGTH.

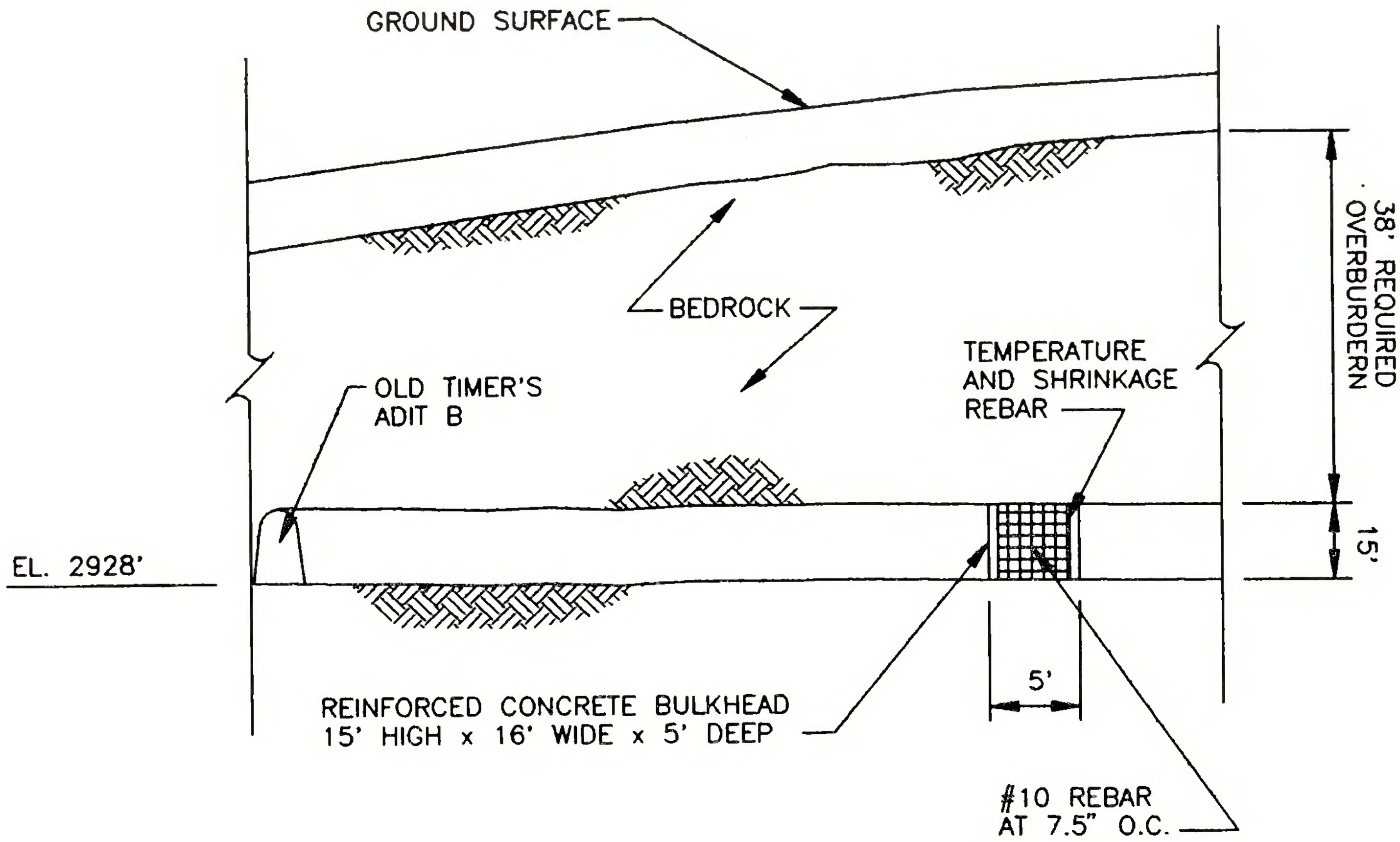
ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1994

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

CONCEPTUAL DIAGRAM OF OLD TIMER'S ADIT A CLOSURE

DATE: MAY 1994 | SCALE: N.T.S. | FIG. NO. 3 1

ECOBAY3B



SIDE VIEW

NOT TO SCALE

NOTES:

1. MAXIMUM HEAD (FT.) 202
2. LINE OF SIGHT DISTANCES FOR ESTIMATING MAXIMUM EARTHQUAKE LOADING APPROXIMATE, BUT CONSERVATIVE. ALL CONCRETE TO BE 3000 PSI COMPRESSIVE STRENGTH.

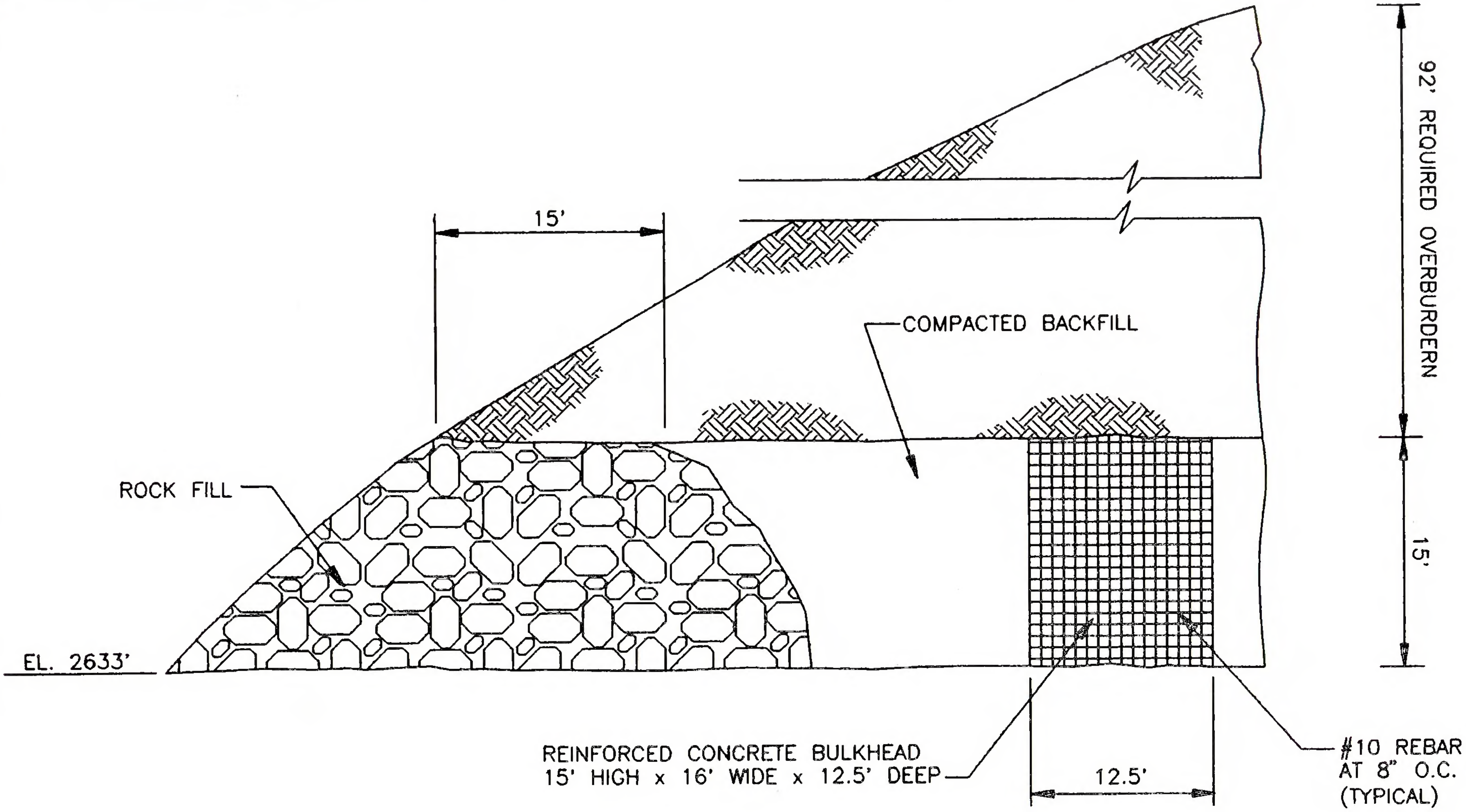
ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1994

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

CONCEPTUAL DIAGRAM OF OLD TIMER'S ADIT B CLOSURE

DATE: MAY 1994 SCALE: N.T.S. FIG. NO. 32

ECOBAY3A



SIDE VIEW

SCALE: 1" = 10'

NOTES:

1. MAXIMUM HEAD (FT.) 497
2. LINE OF SIGHT DISTANCES FOR ESTIMATING MAXIMUM EARTHQUAKE LOADING APPROXIMATE, BUT CONSERVATIVE. ALL CONCRETE TO BE 3000 PSI COMPRESSIVE STRENGTH.

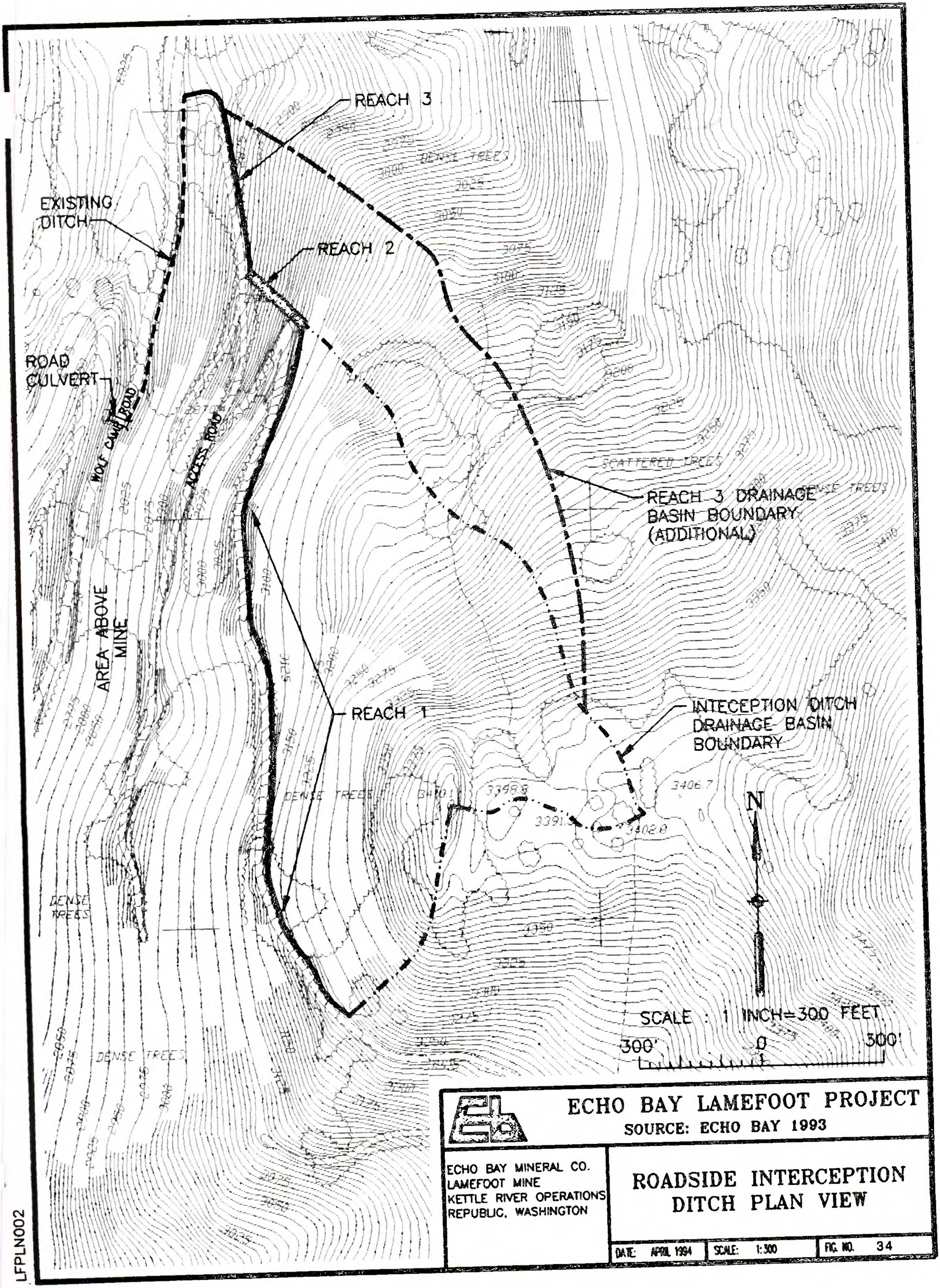
ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1994

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

CONCEPTUAL DIAGRAM OF BACKFILL CROSS CUT CLOSURE

DATE: MAY 1994 | SCALE: 1" = 10' | FIG. NO. 33

ECHOBAY1



ECHO BAY LAMEFOOT PROJECT

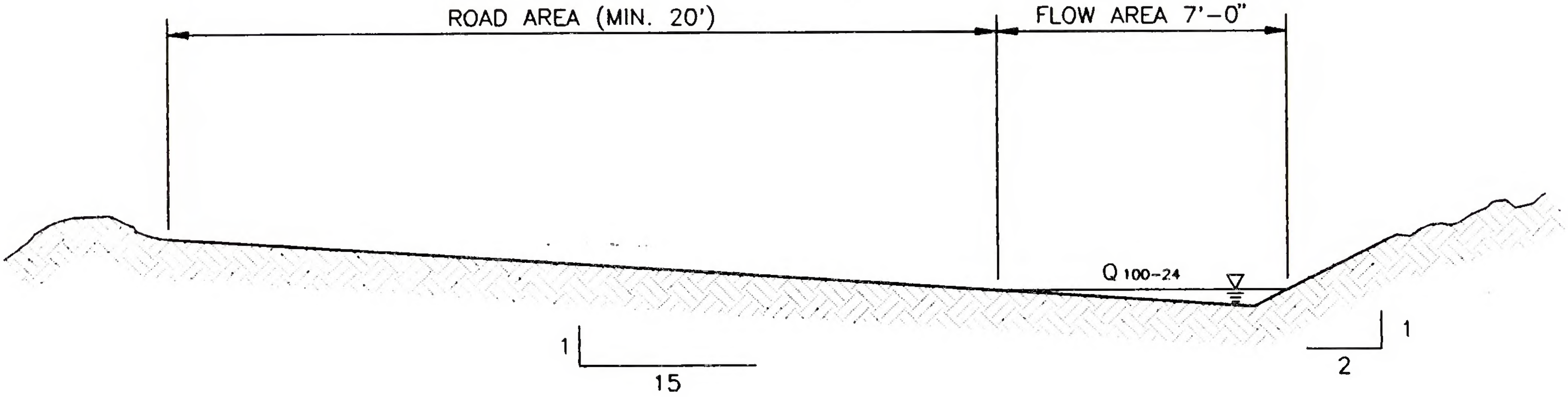
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

ROADSIDE INTERCEPTION DITCH PLAN VIEW

DATE: APRIL 1994 | SCALE: 1:300 | FIG. NO. 34

LFPLN002



KEY FLOW PARAMETERS

SLOPE = 0.12 ft/ft
DISCHARGE (Q_{25-24}) = 6.8 cfs
FLOW DEPTH (d_{25-24}) = 0.4 feet
FLOW VELOCITY (V_{25-24}) = 5.3 fps
FLOW WIDTH (W_{25-24}) = 4.9 feet
DISCHARGE (Q_{100-24}) = 7.8 cfs
FLOW DEPTH (d_{100-24}) = 0.42 feet
FLOW VELOCITY (V_{100-24}) = 5.27 fps
FLOW WIDTH (W_{100-24}) = 7 feet

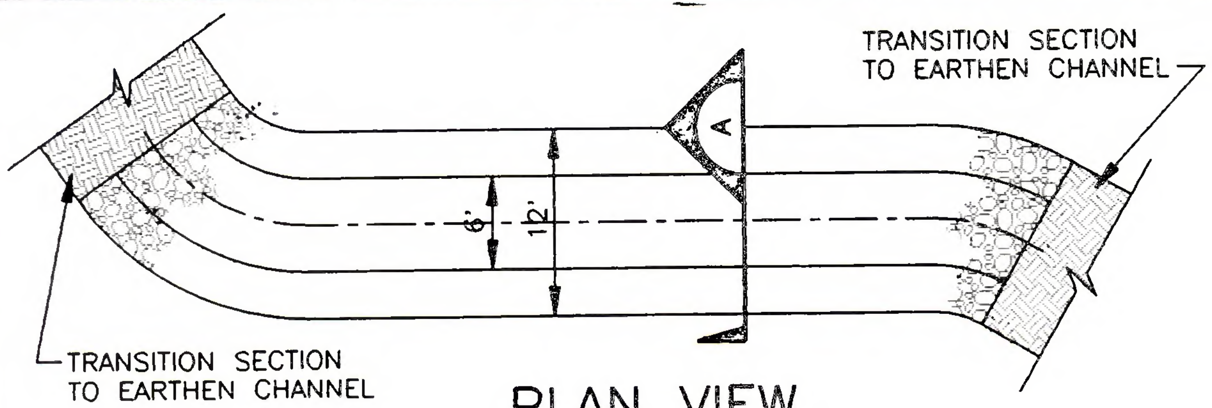
ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

ROADSIDE INTERCEPTOR DITCH – REACH 1

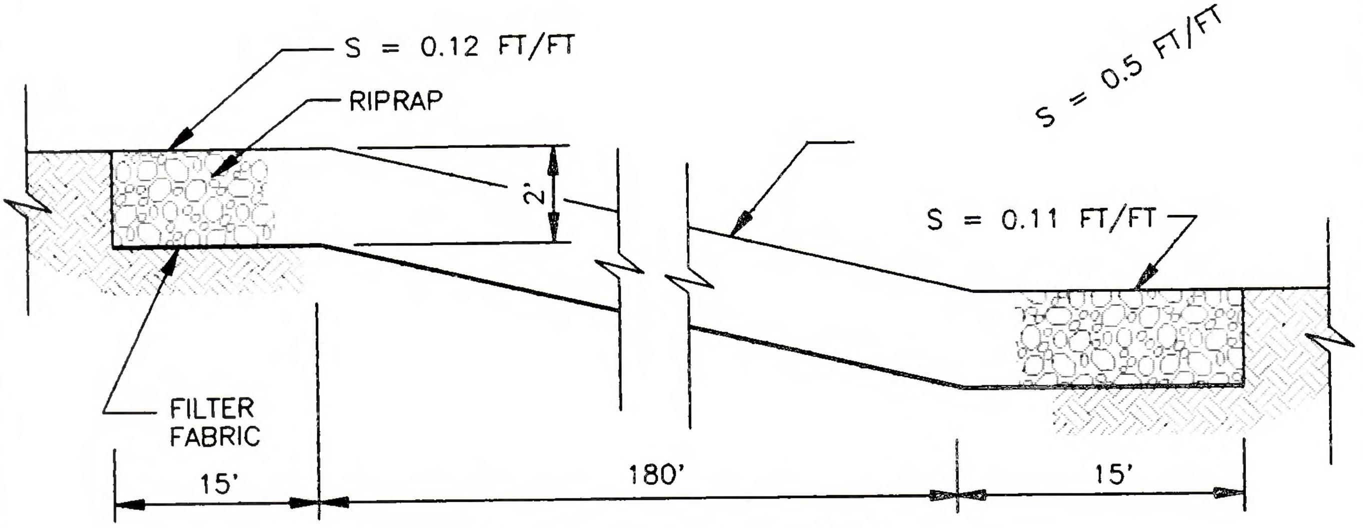
DATE: APRIL 1994 | SCALE: 1" = 4' | FIG. NO. 35

LFREACH1



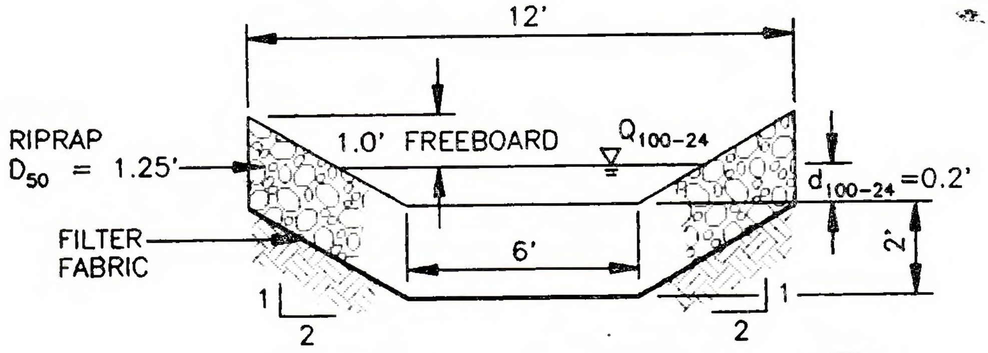
PLAN VIEW

NOT TO SCALE



℄ PROFILE VIEW

NOT TO SCALE



TYPICAL SECTION (A)

NOT TO SCALE

ECHO BAY LAMEFOOT PROJECT

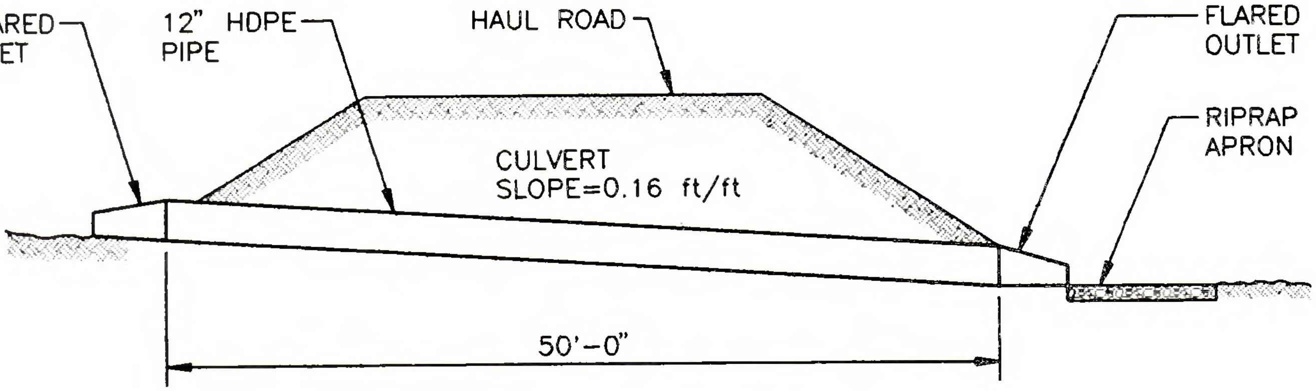
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

ROADSIDE INTERCEPTOR DITCH – REACH 2

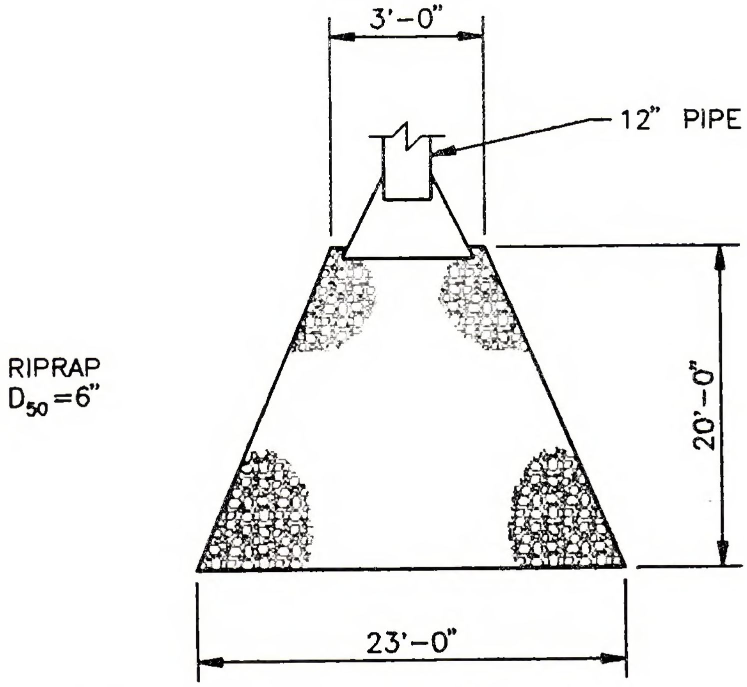
DATE: APRIL 1994	SCALE: AS SHOWN	FIG. NO. 36

23295_1



ROAD CULVERT-CROSS SECTION VIEW

NOT TO SCALE



RIPRAP APRON-PLAN VIEW

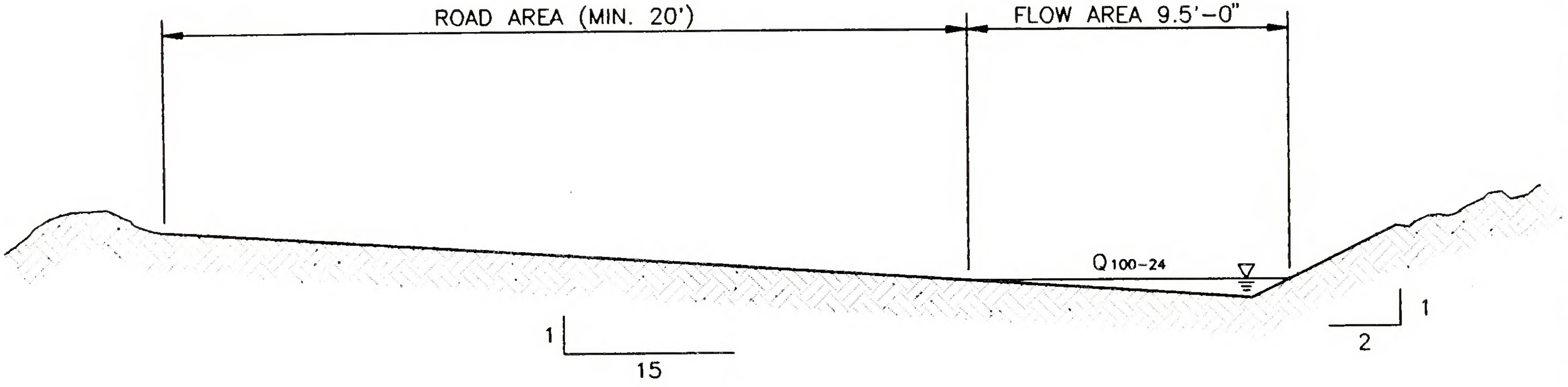
NOT TO SCALE

ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

ROAD
CULVERT
ROADSIDE INTERCEPTOR DITCH
REACH 3

DATE: APRIL 1994	SCALE: AS SHOWN	FIG. NO. 38



KEY FLOW PARAMETERS

SLOPE = 0.11 ft/ft
DISCHARGE (Q_{25-24}) = 9.6 cfs
FLOW DEPTH (d_{25-24}) = 0.5 feet
FLOW VELOCITY (V_{25-24}) = 5.7 fps
FLOW WIDTH (W_{25-24}) = 7.6 feet
DISCHARGE (Q_{100-24}) = 16.6 cfs
FLOW DEPTH (d_{100-24}) = 0.55 feet
FLOW VELOCITY (V_{100-24}) = 6.4 fps
FLOW WIDTH (W_{100-24}) = 9.4 feet

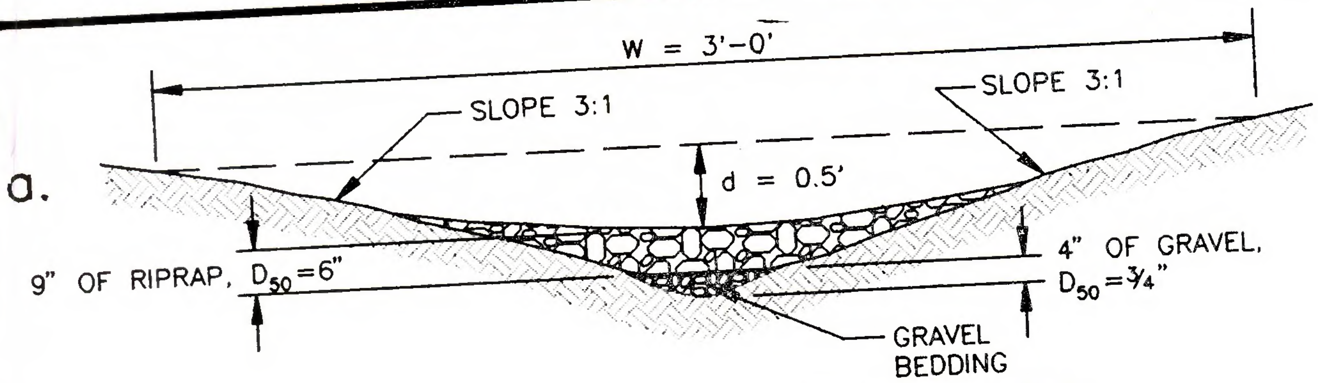
ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

ROADSIDE INTERCEPTOR DITCH – REACH 3

DATE: APRIL 1994
SCALE: 1" = 4'
FIG. NO. 37

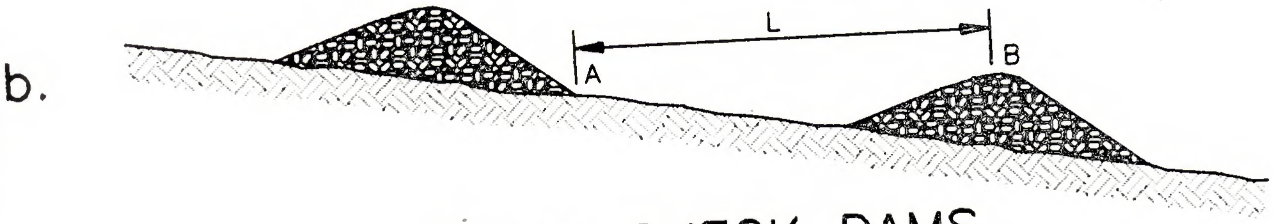
LFREACH3



V-SHAPED WATERWAY WITH STONE CENTER DRAIN

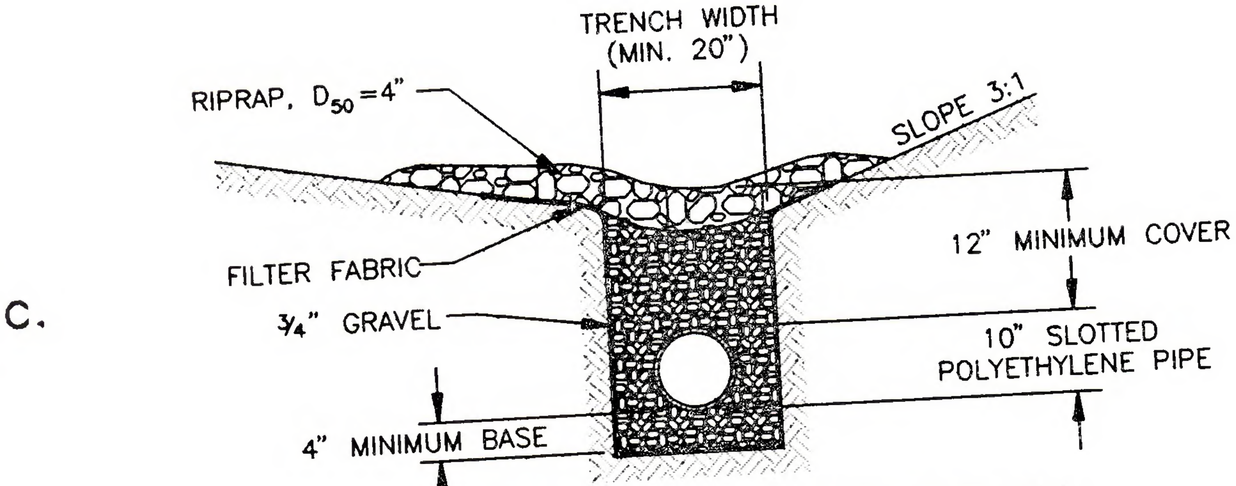
NOT TO SCALE

L = THE DISTANCE SUCH THAT POINTS A AND B ARE OF EQUAL ELEVATION



ROCK CHECK DAMS

NOT TO SCALE



OPTIONAL FRENCH DRAIN (REACH 1 ONLY)

NOT TO SCALE

ECHO BAY LAMEFOOT PROJECT

SOURCE: ECHO BAY 1993

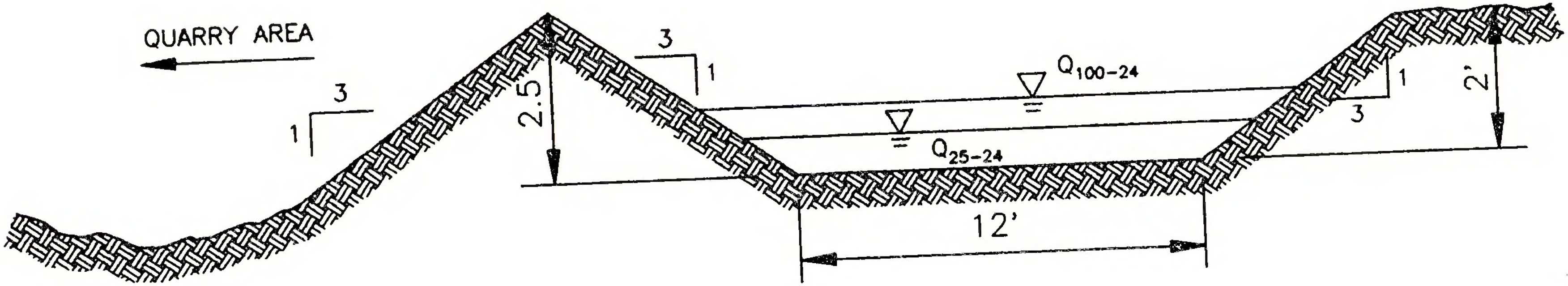
ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

OPTIONAL CHANNEL EROSION CONTROL MEASURES ROADSIDE INTERCEPTOR DITCH REACHES 1 & 2

DATE: APRIL 1994 | SCALE: AS SHOWN | FIG. NO. 39

NOTE:

CONVEYANCE WILL PASS RUNOFF FROM THE 100-YR, 24-HR STORM EVENT IN THE WOLFE CAMP ROAD DRAINAGE AS CALCULATED IN PREVIOUS EIS HYDROLOGY STUDIES (p. 3-7, DEISS).



NOTE:

BERM WILL PREVENT QUARRY AREA RUNOFF FROM ENTERING VALLEY BOTTOM FLOW. AN ESTIMATED 1.3 ACRE FEET OF RUNOFF WILL BE IMPOUNDED AFTER THE 100-YR, 24-HR. STORM.

KEY FLOW PARAMETERS

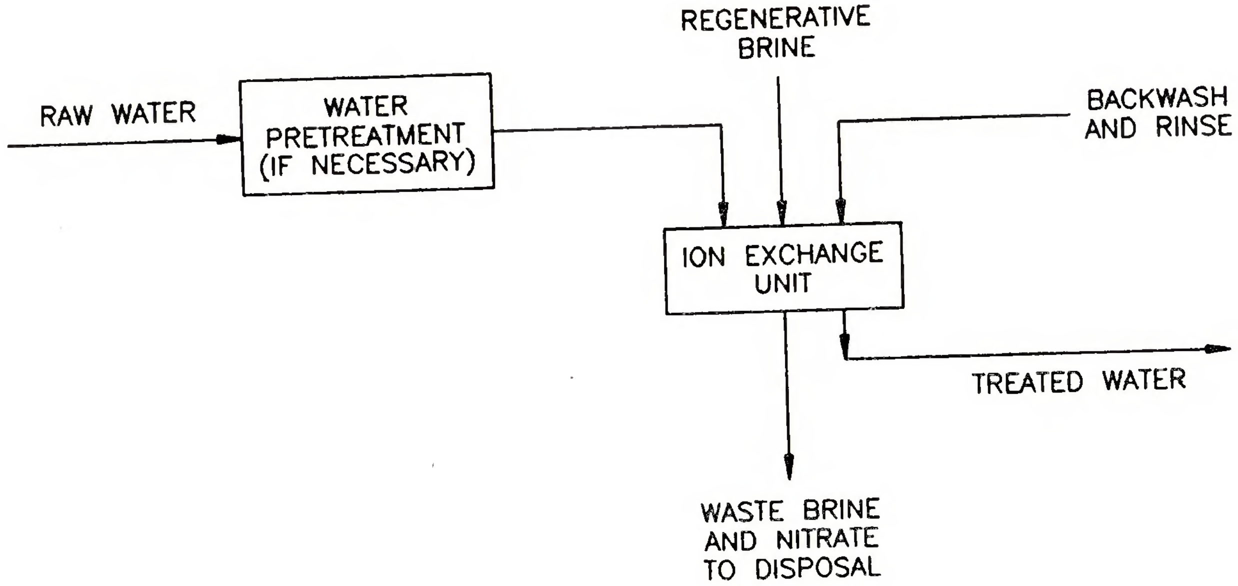
SLOPE = 0.027 ft/ft
DISCHARGE (Q_{100-24}) = 62 cfs
DISCHARGE (Q_{25-24}) = 33cfs
MAXIMUM FLOW DEPTH = 0.77 ft
MAXIMUM VELOCITY = 5.59 fps

ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

QUARRY RETENTION BERM–
WOLF CAMP ROAD DRAINAGE

DATE: APRIL 1994 | SCALE: N.T.S. | FIG. NO. 40



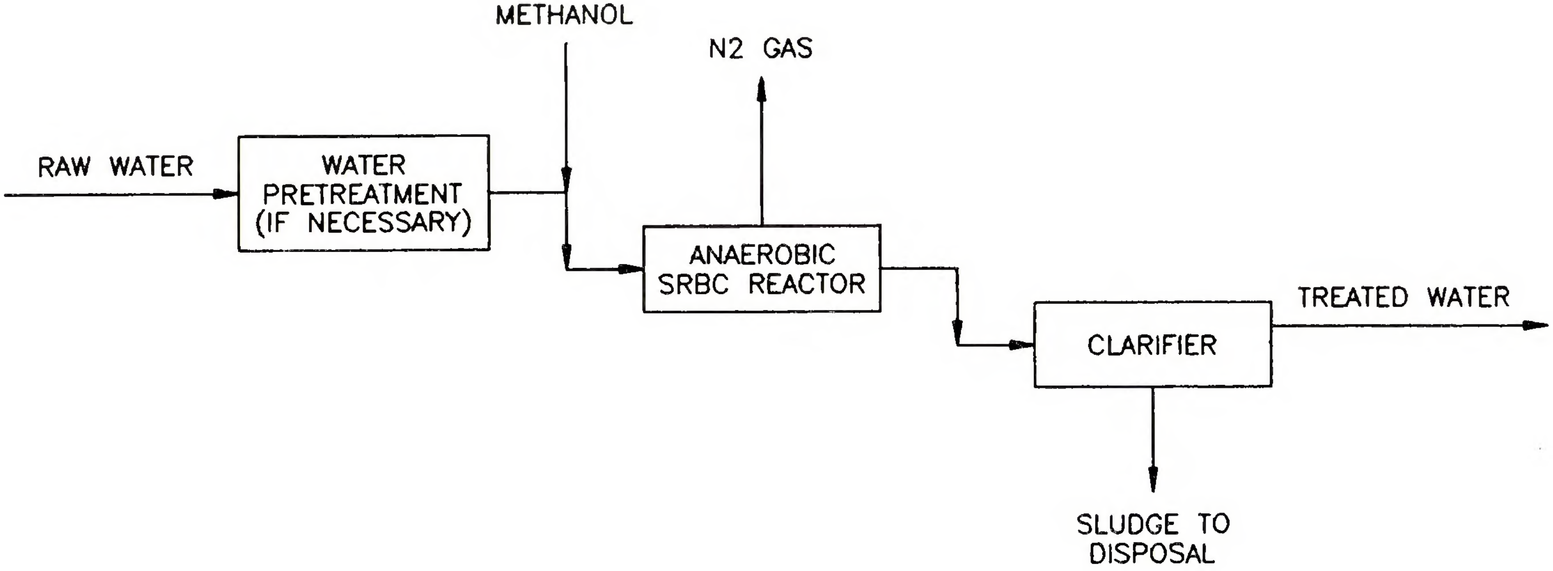
ECHO BAY LAMEFOOT PROJECT

SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

ION EXCHANGE SYSTEM BLOCK FLOW DIAGRAM

DATE: APRIL 1994
SCALE: NONE
FIG. NO. 41



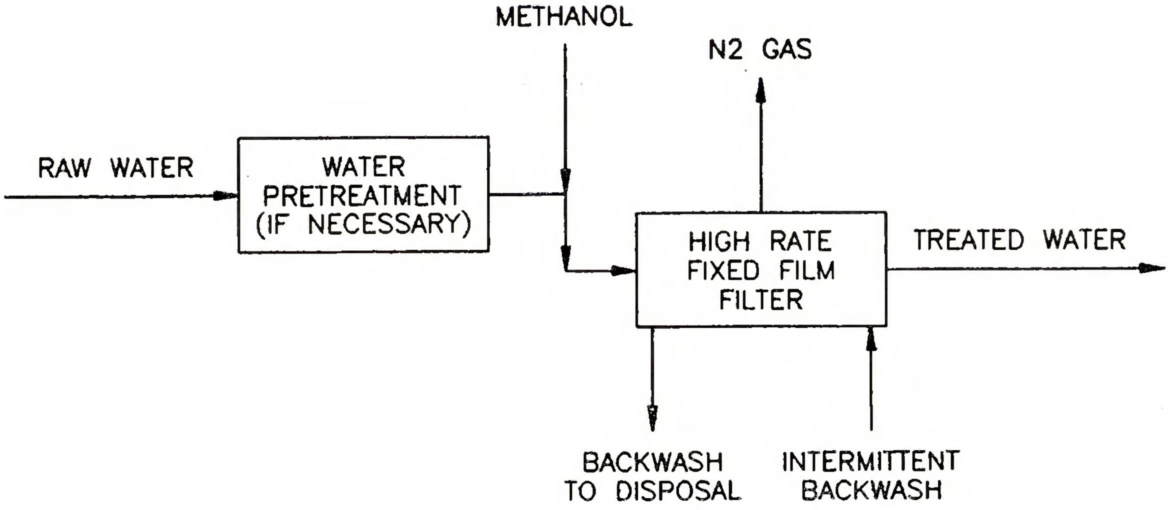
ECHO BAY LAMEFOOT PROJECT

SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

BIOLOGICAL DENITRIFICATION SYSTEM WITH SRBC TECHNOLOGY BLOCK FLOW DIAGRAM

DATE: APRIL 1994
SCALE: NONE
FIG. NO. 42



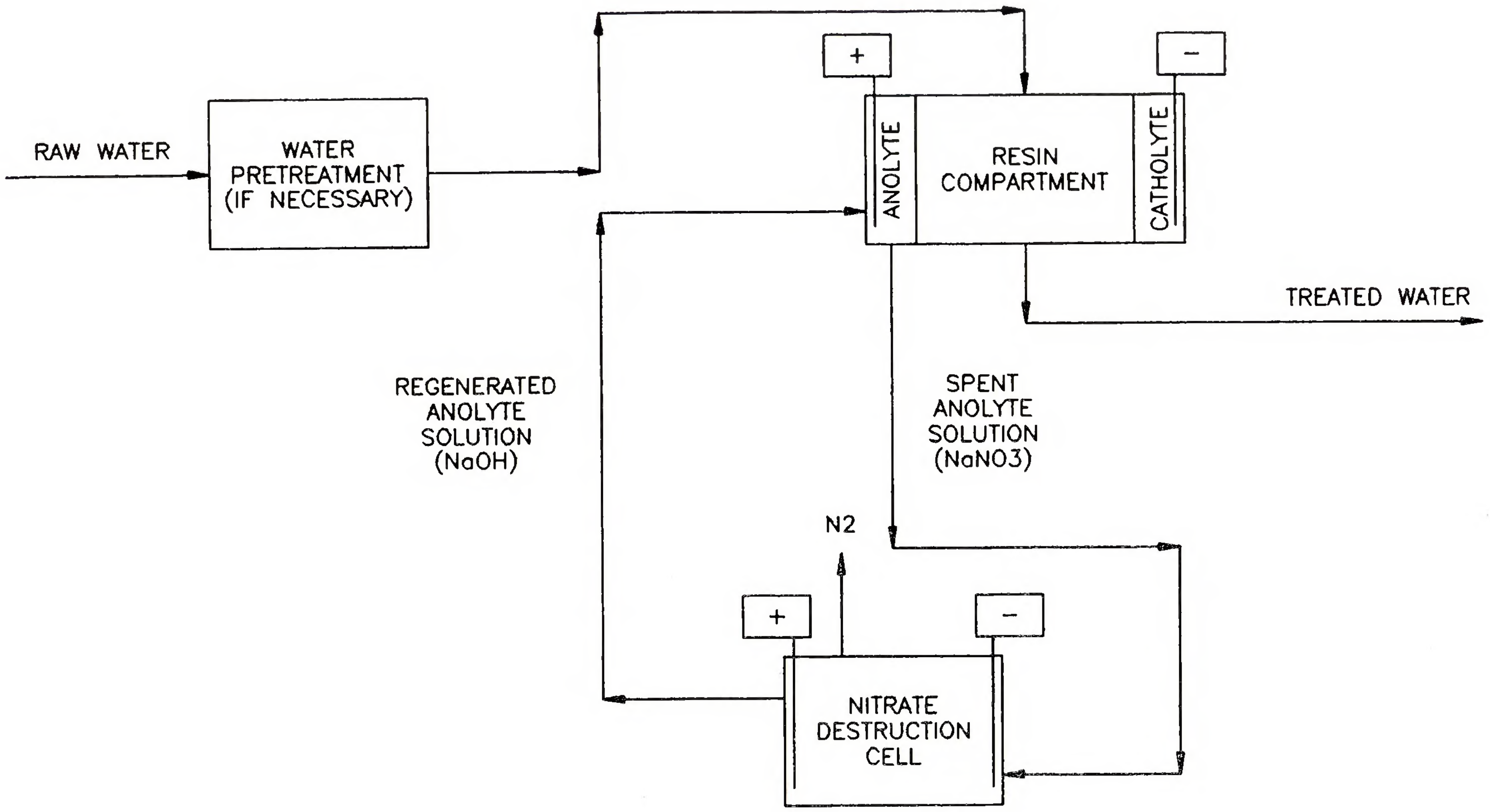
ECHO BAY LAMEFOOT PROJECT

SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

BIOLOGICAL DENITRIFICATION SYSTEM WITH HIGH RATE FIXED FILM FILTER TECHOLOGY INTERMEDIATE BACKWASH BLOCK FLOW DIAGRAM

DATE: APRIL 1994
SCALE: NONE
FIG. NO. 43

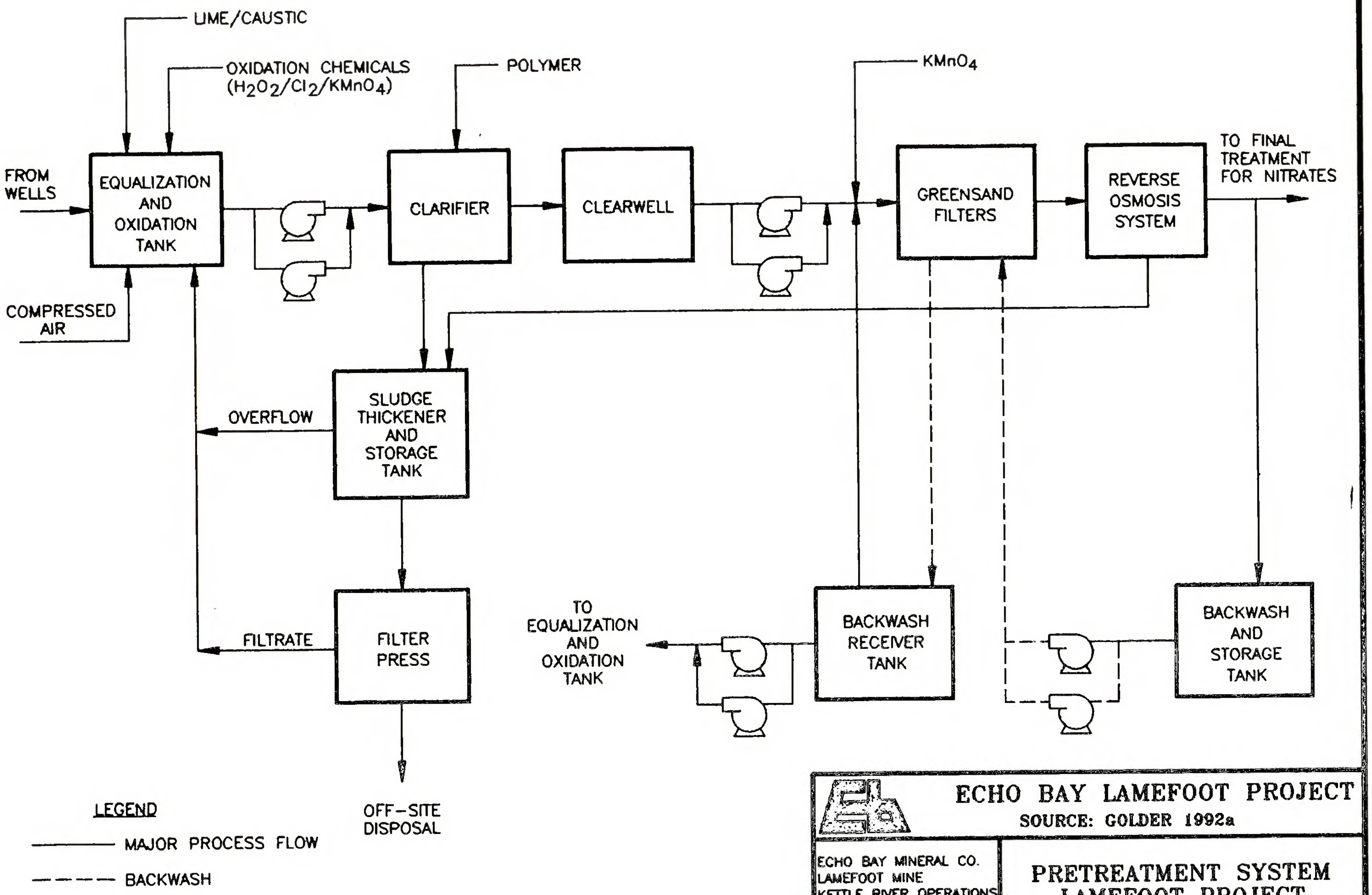

RAW WATER
WATER PRETREATMENT (IF NECESSARY)
+
−
ANOLYTE
RESIN COMPARTMENT
CATHOLYTE
TREATED WATER
REGENERATED ANOLYTE SOLUTION (NaOH)
SPENT ANOLYTE SOLUTION (NaNO3)
N2
+
−
NITRATE DESTRUCTION CELL

ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1993

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

ELECTROCHEMICAL ION EXCHANGE SYSTEM BLOCK FLOW DIAGRAM

DATE: APRIL 1994 SCALE: NONE FIG. NO. 44



ECHO BAY LAMEFOOT PROJECT

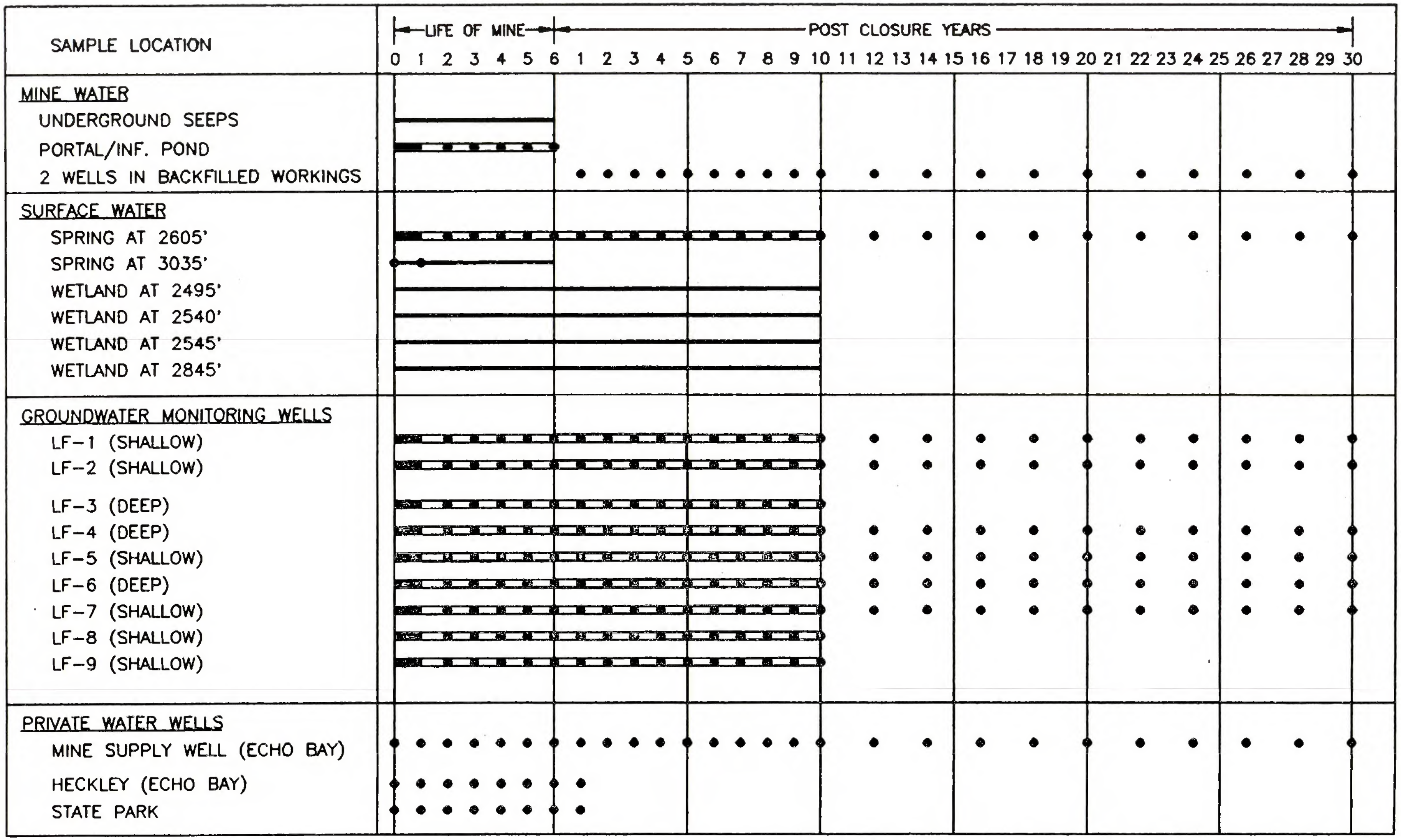
SOURCE: GOLDER 1992a

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

PRETREATMENT SYSTEM LAMEFOOT PROJECT

DATE: JUNE 1994
SCALE:
FIG. NO. 45

X23295_1



EXPLANATION

▬ QUARTERLY OR MORE FREQUENT MONITORING/LONG LIST

▭ QUARTERLY OR MORE FREQUENT MONITORING/SHORT LIST

— QUARTERLY OR MORE FREQUENT MONITORING/FIELD PARAMETERS

● ANNUAL MONITORING/LONG LIST

ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1994

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

LAMEFOOT MINE
WATER QUALITY
MONITORING BY YEAR

DATE: JUNE 1994 | SCALE: NA | FIG. NO. 46

LAME-J3

APPENDIXES

APPENDIX A
KETTLE RIVER MILLING FACILITY

APPENDIX A-1

DESCRIPTION OF MILLING OPERATIONS

TABLE OF CONTENTS

Page

A1.0 KETTLE RIVER MILLING FACILITY 1

A1.1 Milling Process 1
A1.2 Tailings Storage 2

A1.2.1 Pond Design Criteria 2
A1.2.2 Pond Construction Criteria 5

A1.3 Monitoring of Milling Operations 5

LIST OF FIGURES

Figure A-1 Kettle River Mill Flow Chart
Figure A-2 Phase I Tailings Facility and Mill Plant Layout
Figure A-3 Phase IIA Tailings Facility and Mill Plant Layout
Figure A-4 Phase IIB Tailings Facility and Mill Plant Layout
Figure A-5 Geologic Cross-Section Through Current and Planned Phases of Kettle River Tailings Pond

A1.0 KETTLE RIVER MILLING FACILITY

A1.1 Milling Process

The Key mill processes all gold ores produced by the Kettle River Operations. The precious metal extraction method is a typical carbon-in-leach technique commonly used for similar ores throughout the world. The mill design and operation has been approved by the water quality section of the Dept. of Ecology.

The ore is initially transported from a stockpile area near the mill to a crushing circuit housed in an insulated enclosure. Primary crushing is accomplished by a jaw crusher which sizes material to less than 4" in maximum dimension. A secondary cone crusher reduces the size to 3/4". A conveyor transports the crushed material to a fully enclosed screening plant which ensures that material larger than 3/4" is recycled for further secondary crushing.

After screening, the crushed rock is pulverized by a rod mill and ball mill circuit which reduces the material to 80% minus 230 mesh. This finely ground ore is then oxidized and leached in a series of tanks in the presence of a cyanide bearing solution. These tanks are located within a cement sump with adequate capacity to hold the contents of a tank should a leak occur.

Gold is dissolved by cyanide in the leach tanks and is adsorbed onto activated carbon. The carbon is then transferred to an enclosed vessel in which the gold is solubilized. Zinc dust, added to the gold bearing solution forms an intermetallic compound between the zinc and gold. This solid is filtered and smelted in the presence of fluxes inside propane heated furnaces. The resulting molten metal, when separated, is composed primarily of gold and silver. This dore' is then formed into bars which are sold for further refining. The slag composed of glasses of flux is sold for its gold content. Carbon is "washed" in a nitric acid solution to remove impurities added by the adsorption process and is then regenerated in an electric kiln. The carbon is then reused. Nitric acid is neutralized and the solution is pumped to the tailings pond.

The leached ore contained within the cyanide bearing process solution is treated in a cyanide destruction plant utilizing the patented Inco SO_2 system. The cyanide is converted to inert nontoxic substances through the process of oxidation.

The current power requirements at the mill/administrative complex are approximately 45 Kw hr per ton of ore. Therefore, based on previously discussed production rates at Lamefoot, power requirements at the mill associated with that ore would be 55,000 to 80,000 Kw hrs/day.

A power line runs from the Bonneville Power Trunk line to the mill site carrying 115 kv. A substation reduces voltage to 12.5 kv then to 4.15 kv. transformers step voltage down to 480 v. Another substation at the mill reduces voltage from 12.5 kv to 575 v, the voltage of mill components.

A schematic of the mill process is shown on Figure A-1 and the mill plant layout is shown on Figure A-2.

A1.2 Tailings Storage

The treated solution and leached ore (tailings) are stored in a permanent tailings pond (Figure A-2). The tailings pond design has been approved by the Washington DOE. Pond chemistry and monitoring are subject to restrictions specified in a waste water discharge permit issued by the water quality section of the DOE which are shown in Appendix E.

A1.2.1 Pond Design Criteria

The existing pond design is addressed in a Design Criteria Summary prepared in 1988 by Golder Associates (1988). The pond is a zero discharge facility utilizing a fully lined design with underdrain. The liner is composed of very low density polyethylene (VLDPE). The liner rests on a bed of permeable sand which overlies a network of perforated PVC piping. This underdrain serves to capture any natural groundwater which might percolate under the liner from the perimeter of the impoundment. The underdrain also acts as a leak detection system so that any cyanide bearing waters from the pond which escape the liner can be detected. All waters caught by the underdrain, regardless of composition, are pumped into the lined tailings pond.

Figure B-9 from the FEIS (DOE, 1988) shows the location of the existing tailings facility and the subsequent expansions to begin in 1993.

The expansion of the existing tailings pond will be constructed in 1993 through 1995. This expansion is fully permitted under the administrative authority of the DOE, Water Quality and Dam Safety Sections. Details of the expansion design are found in the 1992 Design Criteria Summary prepared by Golder Associates (1992).

The first phase (Phase IIA) of the tailings pond expansion consists of the construction of a new pond (cell) located at the western edge of the existing Phase I facility (Figure A-3). A second phase (IIB) will be constructed in 1994 or 1995 (Figure A-4). Phase IIB adds a lift resulting in an increase of the height of the embankment surrounding the existing pond and the Phase IIA. This two phase expansion will ultimately increase the capacity of the pond by 171% over the existing facility.

Phase I of the pond has adequate capacity to hold all of the ores previously mined at the Kettle, Overlook, Key East and Key West mines, as well as the remaining reserves at the currently producing Key East and Key West deposits. In 1993 development ore produced at the Lamefoot deposits will be mined and tailings will be stored in the existing pond.

At current milling rates the pond will reach capacity early in 1994. Consequently, pond enlargement is required in order to accommodate ore from the Lamefoot deposit scheduled to become available in commercial quantities at the end of the first quarter of 1994. Construction of Phase II therefore is required during the construction season of 1993.

Each phase of the pond is designed to withstand specific catastrophic flood events. Freeboard (the unused capacity of the pond after filling) is engineered to accommodate a storm with a recurrence interval of 500 years. The design of each phase of the pond allows for the volume of that storm event to be stored without discharge.

Additionally, the design must withstand storm events which could cause significant damage to the integrity of the dam itself. Design criteria of the embankment address a storm event with a statistically calculated frequency of recurrence of 100,000 years.

Justification of design criteria to conform to the modelled flood events is found in the Design Criteria Summary Reports for Phases I and II.

The design of Phases IIA and IIB generally use the same criteria as the existing pond; however, a composite liner system will be used in the expansion instead of the single liner design of the existing pond. An additional layer of bentonite will be placed directly underlying the synthetic geomembrane liner in order to provide an additional barrier to leakage. This bentonite layer in direct contact with the liner expands on contact with water. Small leaks, which occur in all liners are immediately sealed by this liner. This design exceeds minimum criteria required by the DOE.

A third line of defence against leakage is afforded by the tailings itself. The very finely laminated high density tailings produce anisotropic conditions with low vertical permeability. The fine grained tailings deposit itself acts as an effective low permeability seal in addition to the geomembrane liner. This was demonstrated at the existing impoundment during the initial deposition of tailings. Small leaks in the geomembrane were effectively sealed as tailings covered the liner. Dessication and compaction of the tailings in the vicinity of any liner imperfection assures a long-term seal.

The existing surface water diversion ditch upgradient from the tailings pond and mill site will be extended to intercept the catchment above the Phase II tailings pond expansion. The design and location of this water diversion structure is described in the Phase I and Phase II Construction Design Criteria Summaries. This diversion ditch prevents most surface water from the catchment area above the pond from migrating under the liner. This enhances the sensitivity and effectiveness of the underdrain as a leak detection system.

A1.2.2 Pond Construction Criteria

Pond construction criteria are detailed for the existing facility in a Technical Specifications report by Golder Associates (1988) and a report for the Phase IIA and IIB expansion project (Golder Associates, 1993).

Construction of the pond expansion and installation of the liner will be executed by outside contractors. Quality control and assurance will also be undertaken by a contractor to ensure that specifications required by the DOE are rigidly adhered to. Dam Safety and Water Quality representatives of the DOE will inspect the pond during construction.

The embankment will be constructed of native materials sorted by grain size and placed in lifts for compaction. Construction will occur during the summer months so that proper compaction can be achieved.

Surface water will be diverted from the site by extension of the existing ditch above the pond to the north. No discharge of storm waters from the site to surface waters will be allowed during construction.

Surface (storm) waters from the mill site are channelled to a sump where it is pumped into the tailings pond.

A1.3 Monitoring of Milling Operations

The amended Kettle River Waste Water Discharge Permit defines a monitoring program for the tailings pond and for groundwaters down gradient from the pond. The permit establishes a limit of a monthly average of 40 ppm weak acid dissociable cyanide in the pond, a level below toxicity limits for mammals. This level is readily achieved by use of the current Inco SO_2 cyanide destruction system. Pond waters are sampled for cyanide daily and the results are reported to the DOE monthly.

The underdrain to the pond is also sampled daily. The low flow of groundwater in the underdrain assures that it acts as a very sensitive leak detection system. Several short lived episodes of detectable cyanide concentration in the underdrain have been noted in the history of the pond. The small leaks causing these events healed themselves naturally as discussed in Section A1.2.1. One previously unknown 0.05 gpm leak in a tailings line at the edge of the pond was effectively discovered by slightly elevated underdrain values. This leak of treated low cyanide waters was consequently identified and repaired. The daily environmental monitoring and highly sensitive analytical limits allowed for early detection and repair of this small leak. As stated previously, all underdrain waters are collected and routed to the tailings pond.

Two groundwater monitoring wells are located directly down gradient from the pond less than 500 feet from the embankment (Figure A-2). Prior to construction several monitor wells were installed to monitor water at the bottom of the unconsolidated gravels at the mill site.

After the pond was constructed these wells went dry. This is because the upgradient catchment area had been affected by the pond itself. The diversion ditch above the pond prevented surface water from reaching the glacial gravels and filtering downward to bedrock. Similarly, the covering of most of the remainder of the catchment with the liner further reduced the groundwater source. Finally, the underdrain catches much of the remaining groundwater and pumps it to the pond.

A drilling program was initiated in 1991 to locate water to monitor down gradient from the pond. After several holes were drilled, a small amount of groundwater was located at the top of a silty horizon in the glacial gravels. Two holes were developed to monitor water at this horizon.

The small amount of groundwater downgradient from the pond enhances the sensitivity of these monitor wells as a leak detection system.

No abnormal chemistry, including detectable cyanide, have been observed in the groundwater monitoring program since inception of the project.

No violations have ever been issued by the Department of Ecology regarding any aspect of mill or tailings storage operations.

A new groundwater monitoring well will be installed down gradient of the Phase IIA expansion as shown on Figure A-5. Also shown are the locations of existing monitor wells and the underdrain monitoring site.

APPENDIX A-2
EXISTING ENVIRONMENT - MILLING SITE

TABLE OF CONTENTS

Page

A2.0 EXISTING ENVIRONMENT - MILLING SITE 1

A2.1 Hydrology 1

A2.1.1 Surface Waters 1
A2.1.2 Groundwater 1

A2.0 EXISTING ENVIRONMENT - MILLING SITE

A2.1 Hydrology

The site is located in a semiarid region, receiving 16 inches of precipitation per year. Of this amount, about half occurs as snowfall from November to March. The watershed above the mill and pond (including the Phase II-B expansion) covers 184 acres. The watershed is characterized primarily by rocky, grassy slopes with limited stands of spruce and tamarack.

A2.1.1 Surface Waters

No surface water bodies are present on the site. Runoff from the site and the watershed above it generally enters the North Fork Sanpoil River located immediately to the south. Runoff from the slopes above the tailings pond and mill is collected by a diversion ditch, sized to handle a 100 year storm. The ditch is routed to the east and discharges off site on the slope above the North Fork Sanpoil River. Runoff from the outboard slopes of the pond is limited and occurs as sheet flow, eventually finding its way to the North Fork Sanpoil River.

All precipitation and runoff which comes into contact with the tailings and process water is retained in the tailings pond. Runoff from the mill area is channeled to a sump and then pumped into the tailings pond.

A2.1.2 Groundwater

The soils on site are a complex series of alluvial deposits, probably deposited in a braided stream environment. Some lake sediments are also interbedded with the alluvial sediments, indicating periodic damming of the North Fork drainage by ice in the main Sanpoil valley. A layer of lodgement till is present below the terrace deposits along the outer limits of the upper terrace, but pinches out closer to the center of the valley. The bedrock in the area consists of Tertiary volcanics. A stratigraphic cross-section through the Phase I pond is shown on Figure A-2.

The alluvial materials are predominantly sand and gravel with some cobbles. However, they range from very coarse materials with boulders up to two feet in diameter, to silty sand. The materials were encountered in somewhat discontinuous and lenticular units distinguished by differences in grain size distribution. The lake deposits consist of silt, interbedded with fine sand and clay.

Groundwater in the area flows predominantly along the bedrock contact and within the bedrock fracture system. The discontinuous lake deposits also form local aquitards resulting in a number of localized perched water tables within the soil overburden.

Two groundwater monitoring wells were installed down gradient from the Phase I pond prior to construction of the pond. These wells were completed at the soil/bedrock contact. Baseline water samples were obtained from these wells. After construction of the pond, the monitoring wells went dry, likely the result of covering some 35 acres of the contributory infiltration area with an impermeable liner.

Two replacement wells were installed in October of 1991 at the locations shown on Figure A-2. These wells were completed in a perched water table within the soil overburden. The location of MW-1 and the stratigraphy it intercepts is shown on Figure A-5. No abnormal chemistry has been detected in these wells since they were installed.

A third monitoring well will be installed down gradient from the Phase II-A expansion, as shown on Figure A-2, to provide complete coverage of the expanded pond area.

APPENDIX B
OTHER ELEMENTS OF KETTLE RIVER OPERATIONS

APPENDIX B-1
KETTLE MINE

Kettle Mine

The Kettle mine is located west of the town of Curlew as shown on Figure 1. An underground exploration program was conducted from October 1987 to June 1989 which was similar to the program now in progress at the Lamefoot mine. A decline ramp was driven into the footwall of the deposit east of the steeply dipping vein system (Figure B-1). From this ramp core holes were drilled to define the veins which were discovered by surface drilling. Crosscuts to the veins and drifts along the veins were part of this underground program.

After full delineation of the orebody was completed the veins were mined utilizing long hole open-stoping as the primary mining method as described in Section 3 of the Lamefoot Mine. Commercial production occurred between June 1989 and May 1990. Ore was hauled via the West Kettle River Road to highway 21 then south by the same route proposed for the Lamefoot ore haul. The production rate averaged about 300 tons per day based on a seven day per week basis. Actual mining took place both on five and seven day per week schedules.

The ore haul varied from 200 to 600 tons per day, five days per week. Ore trucks were of 20 ton capacity and hauled 10 to 20 ton tandem trailers depending on availability and road conditions.

The geology of the deposit mined at the Kettle operation is similar to veins exploited in the Republic District in the vicinity of the town of Republic. The veins consist of quartz with minor (up to 3%) pyrite. Gold occurred as fine disseminations of electrum. Trace amounts of pyrargyrite also occurred in the veins. Unlike the deposits in the Republic District, high concentrations of selenium were not present.

The wallrock to the veins at the Kettle Deposit consisted of Tertiary age Sanpoil volcanic rocks of rhyodacite composition. Phyllic alteration occurs near the veins which is characterized by the presence of sercite and carbonate alternation products and 1-3% pyrite.

The total amount of ore mined at the Kettle deposit was 24,600 tons. Waste rock removed was deposited in a permanent storage area totalled 120,000 tons.

During 1993 the waste rock storage area will be contoured and topsoil will be placed over it. Revegetation will consist of a grass mixture which will result in the re-establishment of native species.

Many of the drill roads on the surface above the Kettle mine have been recontoured and revegetated with grasses. Others have naturally revegetated over time. During 1993 and 1994 the only maintenance of surface drill roads will be weed control by the application of the herbicide Tordon.

The mine itself is currently temporarily closed. No economic ore remains, however, exploration potential may still exist near the workings so permanent closure may not occur until ultimate Kettle River reclamation.

Since the access to the Kettle deposit is a decline and the phreatic water level in the area is below the portal, no discharge of mine waters will occur as a result of closure of the Kettle mine.

APPENDIX B-2
OVERLOOK MINE

Overlook Mine

The Overlook mine is located approximately 2 miles east of the Key mill. This underground operation was initiated in September 1988 and was the primary ore source for the mill until the end of 1992. At that time the mine was closed due to the high cost of operation.

Two decline ramps access the orebody which was mined by a combination of longhole open stoping in a room and rib pillar configuration and by room and post pillar methods. The relatively flat configuration of the lenticular shaped deposits at Overlook required backfilling to enhance ground stability. Some of the backfill contained cement so that mining could take place adjacent to placed fill.

An average of about 1500 tons per day was produced by the Overlook mine during full scale production. This ore was hauled on a private unpaved haul road to the mill.

The character of the ore and waste rock of the Overlook mine is the same as those described at the Key and Lamefoot deposits (see: EA; BLM, 1992 and FEIS; DOE, 1993 and FEIS; USFS, 1992).

The total ore and waste rock which were mined at the Overlook deposit were 1.6 million and 245,000 tons, respectively. Approximately 330,000 tons of gold bearing mineralization remains in the mine but is currently uneconomic due to high mining costs. This ore may be exploited if other ore sources are temporarily unavailable and mill feed is required to avoid shutdown. Therefore the mine is under a temporary closure status.

Because the portal is located above the phreatic groundwater level no discharge from this mine will occur after permanent closure.

Reclamation is 90% complete for drill roads on the surface above the Overlook deposit. This reclamation consisted of recontouring and revegetation with native grasses. Reclamation of these roads occurred during 1990, 1991 and 1992. Revegetation has been very successful.

APPENDIX B-3
KEY MINES

Key Mines

The two deposits, Key East and Key West are the primary ore sources for the Key mill through March of 1994. The surface mining activities and reclamation for these operations is described in detail in the Key Expansion EIS (USFS, 1992). Reclamation will begin in late 1993.

The existing environment of the area around and in these deposits is also described in detail in the FEIS.

Approximately 45,000 tons of ore grade material have been recently identified as an underground minable resource which will remain after surface mining is complete. This material may be mined by room and pillar mining method. Access to this ore would be from the pit wall. A portal would be located on private land and the mined ore occurs under both fee land and U.S. Forest Service land.

APPENDIX B-4
K-2 PROSPECT

K-2 Prospect

The K-2 prospect lies on fee land approximately 2 miles west of the Kettle mine. The geology which is similar to the Kettle deposit, consists of a steeply dipping system of low sulfide quartz veins containing several percent pyrite and finely disseminated electrum.

Approximately eighty drill holes have been completed in the vicinity of the deposit defining a strike length of about 1500 feet. Within the most prospective area, vein widths vary from several feet to about 60 feet and the vertical extent appears to be about 500'.

Currently, drill density along the vein is insufficient to define the mineralization as economic. Therefore no reserves are currently reported for the deposit. However, it is likely that an underground exploration program will be initiated in late 1993 or in the spring of 1994 to assess the feasibility of mining the deposit by underground techniques similar to those described at the Lamefoot and Kettle deposits.

Based on current knowledge, the size and geometry of the K-2 deposit suggest that a mining rate of 500-600 tons per day would be possible. However, if additional mineralization is encountered a higher rate may be sustainable.

It is assumed for the purposes of discussion that the deposit will be mined by underground mining methods but further study must be completed in order to confirm this hypothesis.

If underground mining is the chosen method, then a portal location would likely be at the lowest feasible location north of the deposit and south of the west Kettle River road. This portal location will not be located below phreatic groundwater surface as determined from piezometer measurements in drill holes. These measurements will be taken during the summer of 1993.

The portal and site infrastructures, if located north of the deposit, will be situated as to cause the least visual impact practical from the West Kettle River road. If exploration proves to be successful then ore produced would be hauled by highway trucks to Highway 21 via the West Kettle River road and by the same route established for the Lamefoot ore haul and used in transporting Kettle ores.

It is anticipated from preliminary metallurgical studies that any ore produced from the K-2 deposit would be amenable to milling by the Key mill without substantial modification to the milling circuit. Any minor modifications to the circuit which may be required would be determined by actual tests of bulk samples obtained during an exploration program.

The two phase tailings pond expansion currently in progress and previously permitted would be adequate to accommodate the ores from the K-2 deposit. A currently estimated size of 875,000 tons based on widely spaced drilling could be accommodated even if all of the presently indicated Lamefoot resource is converted into reserves.

Prior to startup of an underground exploration program the SEPA process will be undertaken through the Department of Ecology. At that time, preliminary engineering of a proposed underground program and surface site facility will be complete. Also an assessment of the potential for water quality impacts will be prepared.

APPENDIX B-5

OTHER ECHO BAY ACTIVITIES

Other Echo Bay Activities

Echo Bay Minerals currently plans to continue exploring for additional gold deposits throughout the Ferry, Stevens and Okanogan County areas. These activities may include surface prospecting and drilling on private, federal and state lands, both near the known deposits and in new areas not yet identified. These programs will be similar in nature to the surface drilling programs described in the Lamefoot POO (Echo Bay, 1992a) modified as necessary to account for site specific conditions.

APPENDIX C
BLM RECORDATION NUMBERS OF UNPATENTED CLAIMS AT THE LAMEFOOT PROJECT

Unpatented mining claims of the L.F. group located in the BELCHER MINING DISTRICT, Sections 4 and 9, Township 37 North, Range 33 East, Willamette Meridian, Ferry County, Washington -

CLAIM NAME AND NUMBER	BOOK	PAGE	BLM SERIAL NO.
L.F. # 1	MF	215783	ORMC-132216
L.F. # 2	MF	215784	ORMC-132217
L.F. # 3	MF	215785	ORMC-132218
L.F. # 4	MF	215786	ORMC-132219
L.F. # 5	MF	215787	ORMC-132220
L.F. # 6	MF	215788	ORMC-132221
L.F. # 7	MF	215789	ORMC-132222
L.F. # 8	MF	215790	ORMC-132223
L.F. # 9	MF	215791	ORMC-132224
L.F. # 10	MF	215792	ORMC-132225
L.F. # 11	MF	215793	ORMC-132226
L.F. # 12	MF	215794	ORMC-132227
L.F. # 13	MF	215795	ORMC-132228
L.F. # 14	MF	215796	ORMC-132229
L.F. # 15	MF	215797	ORMC-132230
L.F. # 16	MF	215798	ORMC-132231
L.F. # 17	MF	215799	ORMC-132232
L.F. # 18	MF	215800	ORMC-132233
L.F. # 19	MF	215801	ORMC-132234
L.F. # 20	MF	215802	ORMC-132235
L.F. # 21	MF	215803	ORMC-132236
L.F. # 22	MF	217872	ORMC-135120

APPENDIX D
DOE MONITORING REQUIREMENTS FOR KETTLE RIVER OPERATIONS

APPENDIX D
LAMEFOOT PLAN OF OPERATIONS

The monitoring requirements for the Kettle River Operations are defined in the Dept. of Ecology, Permit No. 8033, issued on December 08, 1989 and amended on October 23, 1992. This document defines discharge location, effluent limitations, testing schedules, testing procedures, records handling and reporting protocol.

Also defined in this permit are project decommissioning for the mill, tailings pond closure, and reclamation requirements.

Monitoring locations defined in the permit include the tailings pond, the tailings pond underdrain, infiltration ponds, monitor wells and surface waters. Monitoring of milling operations is discussed in Appendix A-1. Excerpted from the permit are effluent limitations and surface and groundwater parameters and frequencies which are shown in Appendix D-1.

APPENDIX D-1
SPECIAL CONDITIONS OF DOE MONITORING REQUIREMENTS

SPECIAL CONDITIONS

S1. EFFLUENT LIMITATIONS

After date of issuance and lasting through the expiration date of this permit, the Permittee is authorized to discharge: 1) underground mine water from the Kettle River, Overlook, and Lamefoot mines to infiltrations ponds; 2) treated mill tailings to the Key Mill lined tailings pond; and 3) surface water runoff from the Key Project open pit mines to infiltration ponds subject to meeting the following limitations:

EFFLUENT LIMITATIONS

(1) Kettle River U/G Mine - Infiltration Pond

Parameter	Limitation
pH, s.u.	not outside the range 6.0 - 9.0

(2) Overlook U/G Mine - Infiltration Pond

Parameter	Limitation
pH, s.u.	not outside the range 6.0 - 9.0

(3) Key Mill - Tailings***+++

Parameter	Daily Average*
Cyanide***	40 mg/l

(4) Lamefoot Underground Mine - Infiltration Pond

Parameter	Limitation
pH, s.u.	not outside the range 6.0 - 9.0

(5) Key Project Open Pit Mines - Infiltration Ponds

Parameter	Limitation
pH, s.u.	not outside the range 6.0 - 9.0

*The daily average is defined as the average of the measured values obtained over a calendar month's time.

***Cyanide to be measured in the tailings pond (tailings supernatant). Cyanide is analyzed and reported as weak acid dissociable (WAD).

+++No discharge of wastewater or solids from the tailings impoundment is allowed.

S2. TESTING SCHEDULE

The Permittee shall monitor the wastewater according to the following schedule:

(1) Kettle River Mine - Infiltration Pond

Parameter	Sample Point	Sample Frequency[1]	Sample Type
pH**	outlet of settling ponds	3/week	grab
Cond**	outlet of settling ponds	3/week	grab
TSS	outlet of settling ponds	3/week	grab

(2) Overlook Mine - Infiltration Pond

Parameter	Sample Point	Sample Frequency[1]	Sample Type
pH**	outlet of settling ponds	3/week	grab
Cond**	outlet of settling ponds	3/week	grab
TSS	outlet of settling ponds	3/week	grab

(3) Key Mill - Tailings

Parameter	Sample Point	Sample Frequency	Sample Type
pH	pond	5/week	grab
Cyanide	pond	5/week	grab

Parameter	Sample Point	Sample Frequency*	Sample Type
pH	external sump	1/day	grab
Cyanide	external sump	1/day	grab
Conductivity	external sump	1/day	grab

(4) Lamefoot Mine - Infiltration Pond

Parameter[2]	Sample Point	Sample Frequency[1]	Sample Type
pH	outlet of settling ponds	3/week	grab
Cond	outlet of settling ponds	3/week	grab
TSS	outlet of settling ponds	3/week	grab
NO_3 (as N)	outlet of settling ponds	1/month	grab
Copper	outlet of settling ponds	1/month	grab
Zinc	outlet of settling ponds	1/month	grab
Lead	outlet of settling ponds	1/month	grab
Mercury	outlet of settling ponds	1/month	grab

(5) Key Project Open Pit Mines - Infiltration Ponds

Parameter[2]	Sample Point	Sample Frequency[3]	Sample Type
pH	inlet to ponds	3/week	grab
Cond	inlet to ponds	3/week	grab
TSS	inlet to ponds	3/week	grab
NH_3 (as N)	inlet to ponds	1/month	grab
NO_3 (as N)	inlet to ponds	1/month	grab
Copper	inlet to ponds	1/month	grab
Zinc	inlet to ponds	1/month	grab
Lead	inlet to ponds	1/month	grab
Mercury	inlet to ponds	1/month	grab

*Measured if water present

**If pH is outside range of 6.0-9.0 or, if Conductivity exceeds the statistical norm, EP Toxic metals testing must be undertaken.

[1] If no water is being discharged from the settling ponds to the infiltration ponds, then samples shall be collected from standing water in the settling ponds.

[2] Metals shall be measured as total recoverable metals.

[3] Measured if water is being discharged to the infiltration ponds.

8/92 Amendment to Attachment 1

Surface and Ground Water Sampling Parameters and Frequencies for the Key Project Expansion.

Echo Bay shall monitor surface and ground water at the locations noted in the attached figure (B-1) and descriptions according to the following schedule:

Parameter	Surface Water[1]	Ground Water[2]
Field		
pH	M	M
Conductivity	M	M
Temperature	M	M
Static Water Level		M
Laboratory		
Calcium	M	M
Magnesium	M	M
Sodium	M	M
Potassium	M	M
Chloride	M	M
Carbonate (as $CaCO_3$)	M	M
Iron	M	M
Bicarbonate (as $CaCO_3$)	M	M
Sulfate	M	M
Arsenic	M	M
Barium	M	M
Cadmium	M	M
Hexa-chromium	M	M
Total chromium	M	M
Copper	M	M
Cyanide[3]	M	M
Lead	M	M
Manganese	M	M
Mercury	M	M
Molybdenum	M	M
Nickel	M	M
Selenium	M	M
Silver	M	M
Zinc	M	M
Ortho-phosphate	M	M
Nitrate (as N)	M	M
Turbidity	M	M
Total dissolved solids		M
Total suspended solids	M	M

Note: M means monthly sampling for stations SW-12, SW-13, SW-14, and SW-15, SW-16, KW-1, KW-1A (replacement for KW-1), and KW-2. At least quarterly sampling shall be conducted at stations SW-1, SW-2, SW-3, SW-4, SW-7. Metals shall be measured as dissolved metals.

[1] Surface water stations include SW-1, SW-2, SW-3, SW-4, SW-7, SW-12, SW-13, SW-14, SW-15, and SW-16. SW-16 designates the sampling station at the spring immediately down gradient of the waste rock disposal area.

[2] Ground water stations include KW-1, KW-1A (replacement for KW-1), and KW-2. Piezometer locations KW-1B and KW-1C shall only be tested for static water level. After two years of sampling, the department will review this ground water data and may require the installation of additional monitoring wells.

[3] Cyanide is an optional sampling parameter for stations SW-1, SW-2, SW-3, SW-12, SW-13, SW-14, and SW-15, SW-16, KW-1, KW-1A and KW-2.

APPENDIX E
GEOTECHNICAL STABILITY REPORT

APPENDIX E. Geotechnical Stability Report

The following sub-sections address various issues pertaining to the geotechnical stability of the Lamefoot underground mine, and reference the Plan of Operations (POO).

Proposed Underground Mining

Underground mining appears to be the best method of reducing the potential environmental impacts of mining the Lamefoot deposit while extracting most of this relatively small and deep deposit. The potential environmental impacts of the proposed underground Lamefoot deposit occur where mine workings approach and(or) penetrate the ground surface, where mine workings approach or intersect aquifers and where waste dumps placed on the surface. The unavoidable surface penetrations necessary for the operation of an underground mine at the Lamefoot site are the tunnel portals and the ventilation shaft collar. The potential exists at the Lamefoot site for subsidence associated with underground mining to adversely impact the ground surface and ground water. The surface disposal of waste produced by mine development will be a temporary condition alleviated later by using the development waste as backfill used to mitigate adverse subsidence effects.

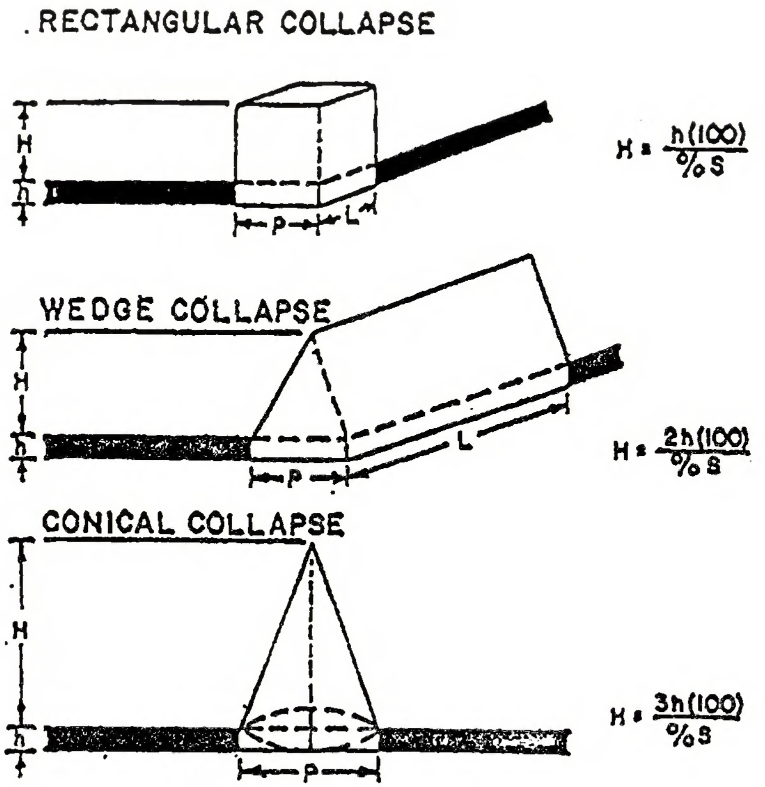
Subsidence

Other sections of the Plan of Operations address selective backfilling of the void produced during mining, to be introduced after stoping is completed. Backfilling of the upper stopes can eliminate the possibility of a collapse chimney breaching the surface, but not the development of trough subsidence at the ground surface. A collapse chimney breaching the surface above one, or more, of the near surface stopes would present both an environmental impact and a safety hazard. The height of the unfilled void between the top of the backfill and the bottom of the surface crown pillars will be carefully controlled in order to eliminate the possibility of a collapse chimney ever breaching the surface over a stope.

Chimney Subsidence

The three upper stopes (stopes 2, 3 and 4) present the potential for developing collapse chimneys through the overlying crown pillars to the ground surface. A 50-foot thick crown pillar was investigated to calculate the maximum allowable height of the unfilled void between the top of the backfill and the bottom of the surface crown pillars over the three upper stopes (numbers 2, 3 and 4). The method utilized was presented by Piggott & Eynon (1977, page 764). This method is presented on Figure E-1.

The maximum height of void partially determines the crown pillar thickness that could conceivably be penetrated by a collapse chimney. The void height must be such that the expansion of the

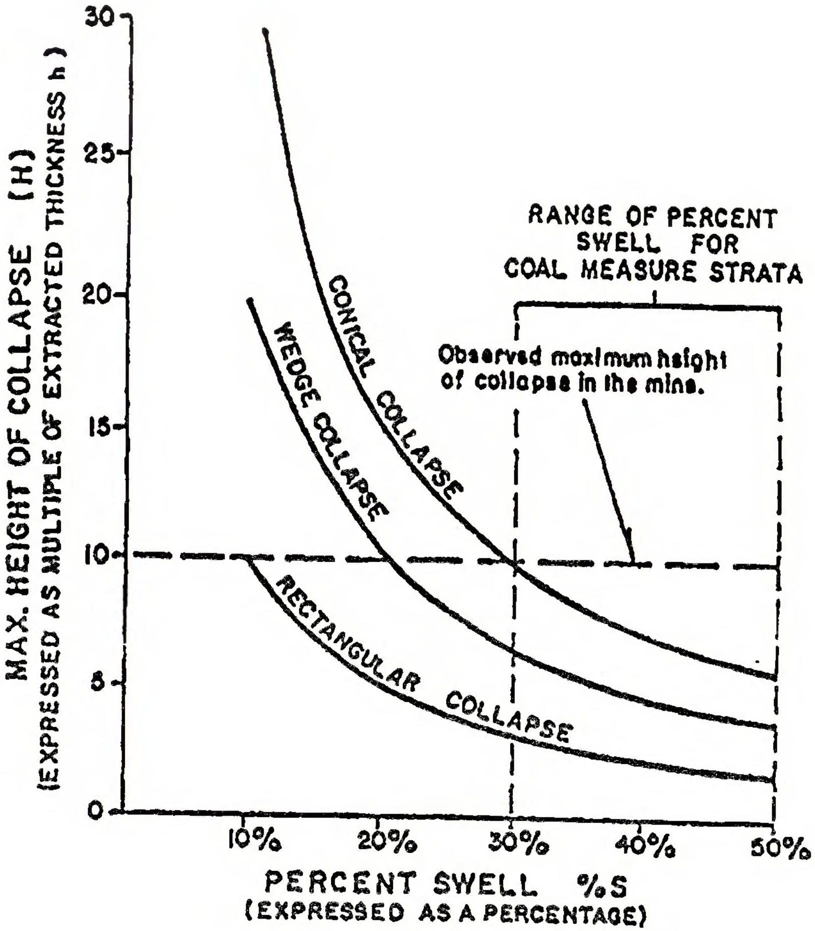


$\%S$ = PERCENT SWELL = $\frac{V_c - V_o}{V_o}(100)$

V_o = ORIGINAL VOLUME OF UNBROKEN STRATA

V_o = VOLUME OF COLLAPSED ROOF BEDS

Diagram showing notation for calculating maximum height of collapse (H) in relation to geometry of collapse. (modified from Piggott & Eynon, 1977)



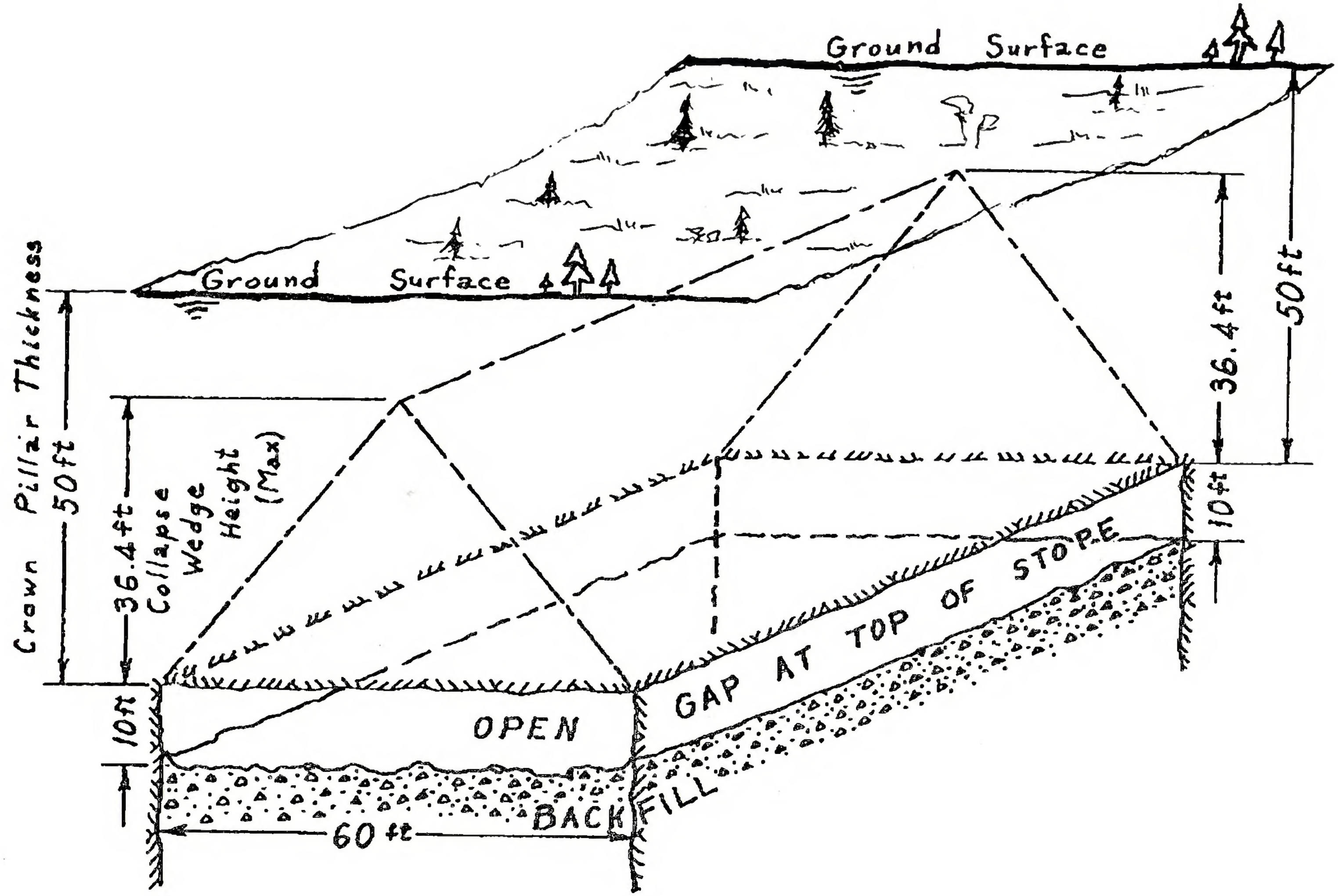
Graph showing variation in maximum height of collapse for different modes of failure and bulking factors. (modified from Piggott & Eynon, 1977)

FIGURE E-1. Prediction of Upward Development of Collapse Chimney collapsing crown pillar rock will completely fill the void before the surface is breached, i.e. eliminate any possibility of chimney collapse breaching the overlying ground surface. The other factors involved in calculating the maximum allowable void height are the percent free swell and the geometry of the collapse chimney. The percent free swell for various rock types has been compiled for haulage equipment by equipment manufacturers and is presented in Table E-1.

A stope that is roughly equidimensional in plan, but whose width is initially self-supporting, will tend to be supported equally on all sides and to develop a conical collapse chimney in the back of the stope, over time. The conical geometry represents collapse toward a single point over the center of the stope. The conical chimney geometry has the potential to penetrate to a greater height above the back of a stope because there is less intact rock to collapse and swell. A stope whose width is less than approximately half its length, but whose width is initially self-supporting, will be supported primarily across its narrowest dimension. Stopes with this plan geometry develop wedge shaped chimneys in the back of the stope, over time. The wedge geometry represents collapse toward a line roughly over the center and paralleling the long axis of the stope. The wedge chimney geometry cannot penetrate as far above the back of a stope because there is more intact rock to collapse and swell. All planned Lamefoot stopes are several times longer than they are wide. Therefore, wedge type chimney collapse should be the type that will develop over the backfilled stopes over extremely long time periods. Openings, all of whose plan dimensions exceed the ability of the overlying rock to span the opening width, produce rectangular collapse chimneys. The rectangular type of chimney collapse geometry penetrates the shortest distance into the rock overlying a stope. Rectangular chimney collapse develops over block caving and longwall mining operations (McNearny & Abel, 1993). All three of these collapse chimney geometries are shown on Figure E-1.

Figure E-2 presents an example case for the maximum wedge-shaped chimney type collapse geometry above a 10-foot void height between the top of the backfill and the base of a 50-foot thick crown pillar. In this example the 55% expansion of rock falling from the crown pillar will stop the upward growth of the collapse chimney at 36.4 feet. Table E-2 presents conservative estimates of the maximum allowable void height, the distance from the top of the backfill to the base of a 50-foot thick crown pillar, that would not breach the overlying surface for both the wedge and conical collapse chimney geometries. At Lamefoot, with its measured 55% expansion of the rock upon breakage, void height should be limited to 13.8 feet under a 50-foot thick crown pillar to eliminate the possibility of chimney collapse breaching the ground surface.

The planned crown pillars vary in thickness along the planned upper Lamefoot stopes, as shown on Table E-3. Therefore, the maximum void height must be limited to the values shown in Table E-3, in



CLIENT/PROJECT
ECHO BAY MINERALS Co.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

TITLE
PREDICTED CHIMNEY COLLAPSE
GEOMETRY FOR EXAMPLE CASE
LAMEFOOT PROJECT

DRAWN
CHECKED
REVIEWED
DATE FEBRUARY, 1994
SCALE
FILE NO.
JOB NO. 23295
DWG NO./REV.NO.
FIGURE: E-2

TABLE E-1. Bank density, swell factor and percent free swell for selected rocks and soils

Rock or Soil	Bank Density	Swell Factor	Free Swell
Basalt	185 PCF	0.67	50%
Bauxite	119 PCF	0.75	33%
Caliche	141 PCF	0.55	82%
Carnotite, primary uranium ore	137 PCF	0.74	35%
Clay, natural	126 PCF	0.82	22%
Coal, anthracite	100 PCF	0.74	35%
bituminous	80 PCF	0.74	35%
Concrete	120-155 PCF	0.72	40%
Conglomerate	153 PCF	0.72-0.63	40-60%
Copper ore, altered siliceous	141 PCF	0.74	35%
Earth, wet	126 PCF	0.79	27%
loam	96 PCF	0.81	23%
Granite, quartzite, fresh	163-170 PCF	0.67-0.56	50-80%
typical values		0.61	64%
Gravel, pit run	135 PCF	0.89	12%
Gypsum	198 PCF	0.57	75%
Iron ore, high grade hematite	174-237 PCF	0.58-0.55	72-82%
limonite	174-237 PCF	0.58	72%
magnetite	204 PCF	0.55-0.58	72-82%
taconite	192-226 PCF	0.58	72%
Limestone	155-163 PCF	0.57-0.60	67-75%
typical values		0.59	69%
Marble, metamorphic	170 PCF	0.57-0.60	67-75%
Montmorillonite, chlorite, kaolin illite, smektite	141 PCF	0.77	30%
Pyrite	189 PCF	0.85	18%
Sand, dry	100 PCF	0.89	12%
damp	120 PCF	0.89	12%
wet	130 PCF	0.89	12%
Sandstone	153-157 PCF	0.60	67%
Shale, mudstone	104 PCF	0.75	33%
Siltstone, hard	153-157 PCF	0.57-0.60	67-75%
soft	126 PCF	0.82	22%
Slate	170-180 PCF	0.77	30%
Traprock, basic, igneous	163 PCF	0.67	50%

Free swell (%) = change in volume broken as a percent of original bank volume

Swell factor = broken density/bank density

Adapted from: Caterpillar, Inc., 1987, Caterpillar performance handbook, p 740 and Euclid Road Machinery Co., 1953, Estimating production and costs

Table E-2. Conservative calculated collapse heights for the hypothetical upper Lamefoot stope geometry, for various free swell percentages and a 50-foot thick crown pillar.

Percent Free	Wedge Geometry	Conical Geometry
30%	7.5 ft	5.0 ft
40%	10.0 ft	6.7 ft
50%	12.5 ft	8.3 ft
55%	13.8 ft	9.2 ft
60%	15.0 ft	10.0 ft
65%	16.2 ft	10.8 ft
70%	17.5 ft	11.7 ft
75%	18.8 ft	12.5 ft

Table E-3. Conservative calculated maximum void height for predicted wedge-shaped collapse chimneys and planned upper Lamefoot stope crown pillar thicknesses and measured 55% swell.

Stope (Zone)	Section	Crown Pillar Thickness	Crown Pillar Width	Maximum Allowable Void Height	Comments
2	N26000	83 ft	26 ft	22.8 ft	Closest to surface
	N26100	99 ft	37 ft	27.2 ft	
	N26200	114 ft	44 ft	31.4 ft	Widest stope span
	N26300	113 ft	37 ft	31.1 ft	
	N26400	113 ft	40 ft	31.1 ft	
3	N26500	64 ft	65 ft	17.6 ft	Widest stope span
	N26600	54 ft	23 ft	14.9 ft	
	N26700	50 ft	26 ft	13.8 ft	
	N26800	56 ft	16 ft	15.4 ft	
	N26900	50 ft	16 ft	13.8 ft	Closest to surface
	N27000	50 ft	16 ft	13.8 ft	Closest to surface
Pillar	N27100				
4	N27200	42 ft	42 ft	11.6 ft	Closest to surface
	N27300	100 ft	100 ft	27.5 ft	
	N27400	104 ft	104 ft	28.6 ft	
	N27500	150 ft	150 ft	41.3 ft	Widest stope span

order to locally eliminate the possibility of wedge-shaped chimney subsidence breaching the ground surface above any planned stopes.

Trough Subsidence

Trough subsidence develops on the surface over underground openings as the result of the deformation accompanying redistribution of the loads previously carried by the extracted waste rock and ore onto the rock remaining over and adjacent to the openings. The magnitude of the trough subsidence over underground openings is dependent on

1. the size of the openings,
2. the depth of the openings,
3. the stresses present and
4. the properties of the rock present.

Surface trough subsidence will be small over the planned near surface Lamefoot stopes because the elastic modulus of rock is high, as shown in Table E-4, and because the high-strength rock in the surface crown pillars, as shown in Table E-5, should be able to support the mining induced stresses. The magnitude of trough subsidence will progressively increase in the case of progressive crown pillar failure, i.e. chimney collapse of the lower part the crown pillar onto the planned underlying backfill.

Boundary element modeling, approximating the geometry and geology of the planned stoping, was performed utilizing more realistic stain softening rock properties. The program employed was MULSIM/NL (Zipf, 1992). The plane-of-the-vein boundary element model consisted of 6650, 20-foot by 20-foot, stope area elements and 430, 100-foot by 100-foot, adjacent boundary elements. The physical property approximations utilized in the boundary element models are presented in Table E-4. The physical properties utilized were based largely on the Lamefoot data presented. The dip of the ANFO Fault controlled orebody was taken as 60 degrees. Figure E-3 presents the case for a uniform 60-foot thick orebody, indicating:

1. the stope layout,
2. the contours of predicted stress normal to the plane of the orebody in the rock adjacent to and between planned Anfo Fault stopes and
3. the rock type distribution reported in the Plan Of Operations (POO).

The rock type distribution follows figures in the POO, with limestone (Pls) present in both the hanging wall and footwall of the southern third of the deposit, with a limestone hanging wall and clastics footwall (Pcl + Pls) in the central section of the deposit and with clastics (Pcl) present in both the hanging wall and footwall of the northern third of the deposit. The two steeply south dipping cross faults were taken from the stoping plans in the POO.

The 3-dimensional plane-of-the-vein boundary element modeling program utilized for this conservative approximation required a

Echo Bay Minerals, Inc.　　Plane of Vein Normal Stress, Strain Softening　　Run No. 1A

Lamefoot Mine Plan, 60-ft Thick, No Backfill

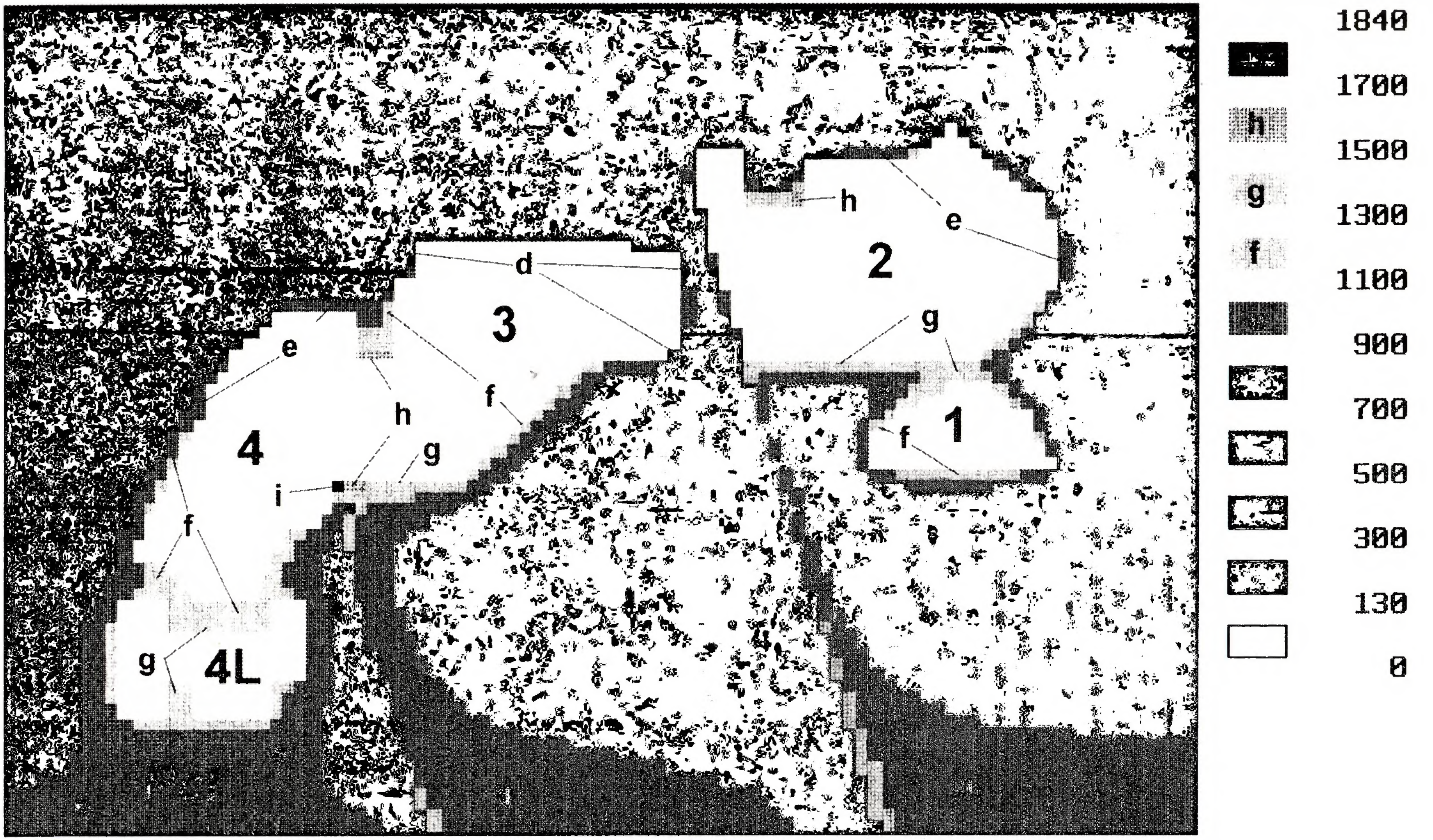


Figure E-3

Table E-4. Physical properties utilized in numerical modeling

Estimation of Unconfined Rock Mass Compression Strength

Unconfined Specimen Compression Strength (Addendum 2, p 2), Limestone Pls - 15,000 psi, and Clastics Pcl - 21,600 psi

Strength Reduction for Size Effect

Widely jointed Limestone, Strength reduction divisor 5.5
(Pratt, et al, 1972)
Limestone mass strength 15,000/5.5 2750 psi

Closely jointed Clastics, Strength reduction divisor 7.5
(Bieniawski, 1968)
Clastic mass strength 21,600/7.5 2900 psi

Estimation of Rock Mass Deformation Modulus

Barton, et al (1974) "Q" Classification for Lamefoot rocks
Limestone = 53, Clastics = 2 (Roberts & Moss, 1993)

Barton, et al (1983) presented relationship between in situ deformation modulus (E) and "Q", as follows:

Mean $= 25 \log_{10} Q$
Maximum $E_{max} = 40 \log_{10} Q$

Limestone mass $= 25 \log_{10}(53) = 43$ GPa $(6.25 \times 10^6$ psi$)$
Limestone mass $E_{max} = 40 \log_{10}(53) = 69$ GPa $(10.0 \times 10^6$ psi$)$
Clastics mass $= 25 \log_{10}(2) = 7.5$ GPa $(1.09 \times 10^6$ psi$)$
Clastics mass $E_{max} = 40 \log_{10}(2) = 12$ GPa $(1.75 \times 10^6$ psi$)$

Bieniawski (1983) presented a relationship between the mean in situ deformation modulus (), "Q" and Rock Mass Rating (RMR), as follows:

RMR = 9 ln Q + 44 and = 2 RMR - 100
Limestone mass RMR = 9 ln(53) + 44 = 80
Clastics mass RMR = 9 ln(2) + 44 = 50

Limestone mass = 2(80) - 100 = 60 GPa $(8.7 \times 10^6$ psi$)$
Clastics mass = 2(50) - 100 = 0 MPa $(0.0 \times 10^6$ psi$)$?

Bieniawski (1983) presented the deformation modulus reduction between laboratory and field determinations for 42 tests. The mean deformation modulus reduction multiplier was 0.457, with a standard deviation of 0.285. Therefore, the recommended deformation moduli (E) for design are, as follows:

Limestone mass E = 0.457 $(8.7 \times 10^6$ psi$) = 4.0 \times 10^6$ psi
Clastics mass E = 0.457 $(1.75 \times 10^6$ psi$) = 0.80 \times 10^6$ psi

Table E-5. Rock strength test results

Rock Type & Number	Compressive Strength (psi)	Tensile Strength (psi)
Limestone		
1	14406	1405
2	15457	1216
7	20811	921
11	14668	2402
12	16372	1367
13	22323	1282
16	16440	712
18	10785	1536
23	16867	830
24	11400	(1)
27	19156	688
29	14735	(2)
37	13698	1115
38	7847	620
40	11537	752
42	13546	1747
Mean	15003	1185
Std. Dev.	3746	495
Coef. Var.	25%	42%

Rock Type & Number	Compressive Strength (psi)	Tensile Strength (psi)
Clastic		
4	19197	2371
5	15230	974
6	13557	1840
9	24183	2638
10	13198	1689
15	23919	1513
20	20871	1072
21	30446	1721
22	23299	942
25	26319	1381
26	12248	(2)
28	27053	2206
30	28003	1552
31	(2)	2167
39	28286	1713
41	17983	1747
44	(2)	1267
Mean	21586	1675
Std. Dev.	6057	494
Coef. Var.	28%	30%

Rock Type & Number	Compressive Strength (psi)	Tensile Strength (psi)
Massive Magnetite		
3	35959	2313
8	16312	1888
14	20919	645
33	47627	1763
34	27886	(2)
Mean	29741	1652
Std. Dev.	12452	712
Coef. Var.	42%	43%

Rock Type & Number	Compressive Strength (psi)	Tensile Strength (psi)
Intrusives		
TBR Dike		
32	20128	1461
35	18727	1989
36	26078	1898
THR Dike		
17	19618	1674
43	14570	(2)
TI Dike		
19	12081	749
Mean	18534	1554
Std. Dev.	4858	495
Coef. Var.	26%	32%

(1) - Bad Test; (2) - No sample

uniform stope width. It was assumed for the initial modeling calculation that the stopes will be 60-foot wide from hanging wall to footwall. A 60-foot thickness is somewhat greater than the average indicated in the cross section presented in the POO. A second model was run approximating the maximum 130-foot mining width. Boundary element modeling was performed for open, unfilled, stopes, for conservatively assumed low-modulus backfilled stopes approximating coarse rockfill without cement and for conservatively assumed high-modulus backfill representing cemented coarse rockfill.

The effectiveness of the backfilling program in limiting the magnitude of trough subsidence of the ground surface overlying the near surface stopes will depend on the actual consolidation characteristics and effective modulus of the backfill actually employed. In the absence of the backfill consolidation characteristics for the actual backfill it was necessary to utilize conservatively low backfill consolidation characteristics to predict stope closure across the plane of the vein. It was conservatively assumed that the magnitude of trough subsidence would equal stope closure. Such an approximation is fundamentally conservative because:

1. total stope closure normal to the plane-of-the-vein must be less than total stope closure down-dip along the much narrower 60-foot and 130-foot vein thicknesses used in the models,
2. total stope closure normal to the plane-of-the-vein must be more than the portion of stope closure from the inward deflection of the hanging wall and
3. there is no record of surface trough subsidence equaling the downward deflection of the hanging wall over the stope.

The results of the boundary element modeling appear reasonable in part because they indicate decreasing amounts of stope closure, as follows:

1. maximum for open stopes without crown pillar failure,
2. intermediate for stopes backfilled with loose uncemented rock fill and
3. the least for cemented rock filled stopes.

Figure E-4 graphically presents stope closure for the stable open stope condition. Table E-6, that follows, presents conservatively predicted maximum stope closure (subsidence), in inches, along critical maximum down-dip stope spans for the three modeled stope types.

The subsidence induced near-surface ground strains were initially very conservatively predicted by assuming that

1. the entire stope closure was transmitted undiminished through the overlying rock to the ground surface and
2. the width of the crown pillar was from the bottom of the crown pillar to the bottom of the stope.

These very conservatively predicted surface subsidence values should considerably exceed those eventually measured by the planned monitoring program, provided the assumptions required are

Echo Bay Minerals, Inc. Plane of Vein Closure, Strain Softening Run No. 1A

Lamefoot Mine Plan, 60-ft Thick, No Backfill

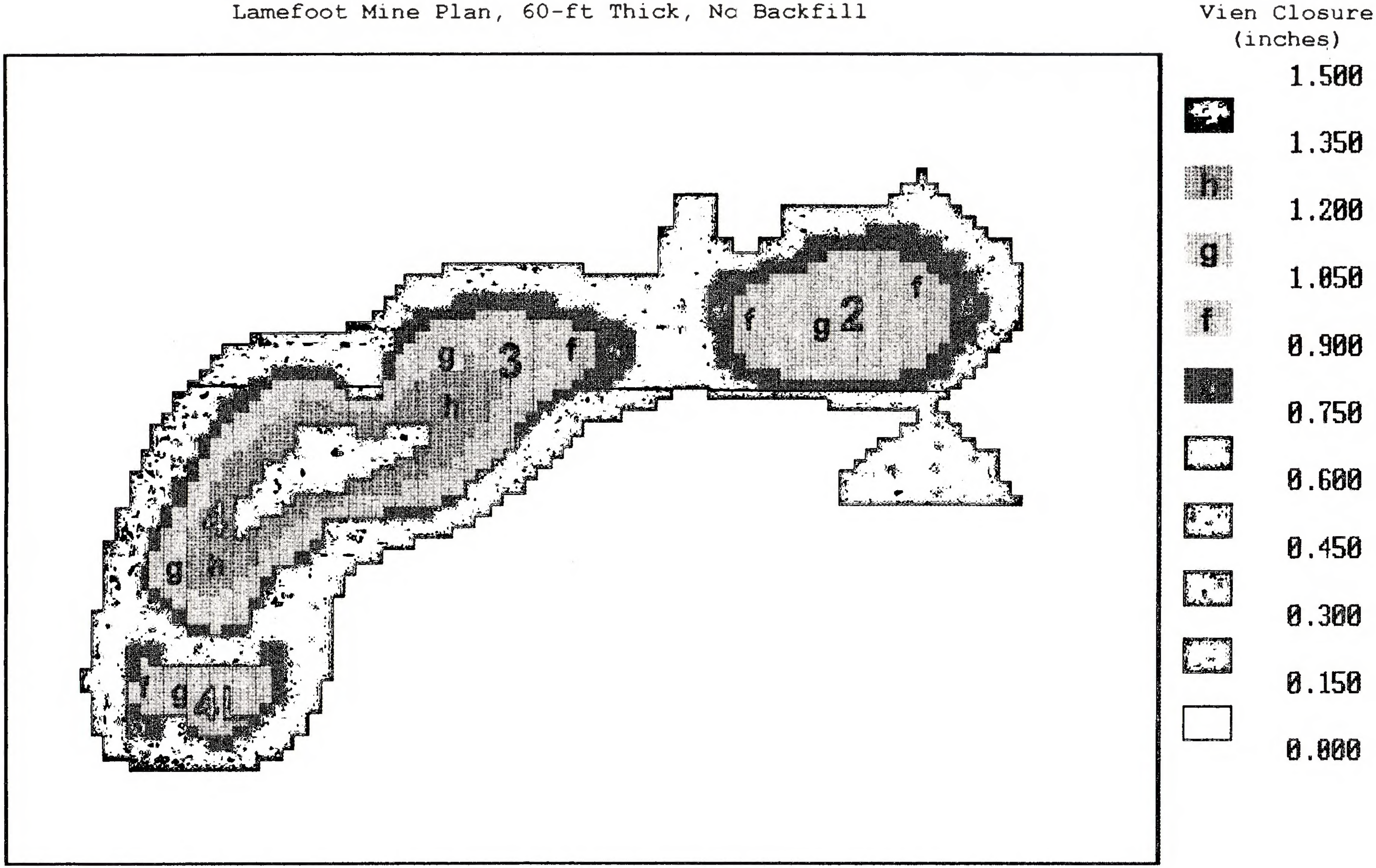


Figure E-4

Table E-6. Maximum possible surface subsidence (inches) above near-surface stopes, conservatively assumed equal to total vein closure and no failure of the surface crown pillar.

Section	Stope Number	Open Stopes No Backfill	Low-modulus Backfill	High-modulus Backfill
		60-foot stope width		
N26090	2	0.95	0.94	0.83
N26930	3	1.35	1.32	1.13
N27250	4	1.37	1.34	1.15
		130-foot stope width		
N26090	2	1.24	1.23	1.15
N26930	3	1.68	1.67	1.52
N27250	4	1.62	1.61	1.42

conservative, as intended. Similarly, the calculated ground surface strains represent local maximum values which should exceed the strains calculated from changes in bay length measured by the surface monitoring net.

Damage to the ground surface or to structures on the ground surface is primarily dependent on the surface strains developed as the result of flexure accompanying surface subsidence centered over the stopes. The maximum surface strains are a function of the maximum surface subsidence (Table E-6), the stope width (60 feet and 130 feet) and the thickness of the crown pillar (50 feet). The maximum surface strains were calculated using the British National Coal Board (1976) subsidence prediction method. These results are presented in Table E-7.

The predicted maximum strains are in the range that would result in the cracking of gypsum plaster, 1000 micro-strain (Voight & Pariseau, 1970, p 736). As such, the maximum predicted surface strains resulting from trough subsidence should produce nearly undetectable cracking of joints exposed in rock outcrops and should not be observable when soil cover is present.

The tensile strain that develops along the outer sides of a subsidence trough will increase the permeability of the rock in the near-surface zone of rock subject to tensile strain (Wardell, 1976, p 7; Kendorski, et al, 1979, p 112; Moore & Nawrocki, 1980, p 58). Babcock and Hooker (1977, p 3) presented guidelines for the magnitude of tensile surface trough subsidence strain which would not result in surface water entering mine workings. They recommend limiting tensile surface strain to 0.00875 in/in (8750με). They referenced British studies where calculated tensile strains of the overlying seabed were as much as 15mm/m (0.01500 in/in; 15000με) without sea water entering the workings. The maximum allowable void height between the top of the backfill and the bottom of the planned crown pillars shown in Table E-3 that would not result in any hydraulic communication between the ground surface and the near-surface stopes is presented in Table E-8. These values are based on limiting surface tensile strains of 8750με and 15000με, for the very conservative assumption that all stope closure results from downward deflection of the crown pillar.

It is not possible to limit the void between the top of the backfill and the base of the crown pillar to the approximately 1-foot indicated on Table E-8. Any gap greater than that indicated in Table E-8 will probably result in a zone of increased permeability near the ground surface and at the outside surface traces of the underlying stopes. Therefore, it will not be possible to mine the upper planned stopes without increasing the vertical permeability of a portion of the rock between the surface and the stopes. The quantity of additional surface water that will enter the upper stopes because of the increased permeability will be minor because of the low magnitude of tensile strain and the

Table E-7. Maximum possible surface subsidence induced strains above near-surface stopes (υε - micro-strain, millionths inch per inch).

Section	Stope Number	Open Stopes No Backfill	Low-modulus Backfill	High-modulus Backfill
		Maximum Tensile Strain (υε) 60-foot stope width		
N26090	2	1030	1020	900
N26930	3	1460	1430	1220
N27250	4	1500	1460	1250
		130-foot stope width		
N26090	2	1340	1330	1250
N26930	3	1820	1810	1650
N27250	4	1760	1740	1540
		Maximum Compressive Strain (υε) 60-foot stope width		
N26090	2	870	860	760
N26930	3	1240	1210	1040
N27250	4	1270	1230	1050
		130-foot stope width		
N26090	2	1050	1050	980
N26930	3	1430	1420	1290
N27250	4	1380	1370	1210

Table E-8. Maximum open gap at top of near-surface stopes with 50-foot thick surface crown pillar with no leakage to stopes (υε - micro-strain, millionths inch per inch).

Section	Stope No.	Lo-modulus Backfill in. (ft)	Hi-modulus Backfill in. (ft)	Lo-modulus Backfill in. (ft)	Hi-modulus Backfill in. (ft)
ASSUMES ALL STOPE CLOSURE RESULTS IN TROUGH SUBSIDENCE					
60-ft stope width					
		8750υε-Limiting Strain		15000υε-Limiting Strain	
N26090	2	7.14(0.60)	7.25(0.60)	12.91(1.08)	13.02(1.08)
N26930	3	6.76(0.56)	6.95(0.58)	12.53(1.04)	12.72(1.06)
N27250	4	6.74(0.56)	6.93(0.58)	12.51(1.04)	12.70(1.06)
130-foot stope width					
		8750υε-Limiting Strain		15000υε-Limiting Strain	
N26090	2	6.85(0.57)	6.93(0.58)	12.62(1.05)	12.70(1.06)
N26930	3	6.41(0.53)	6.56(0.55)	12.18(1.01)	12.33(1.03)
N27250	4	6.47(0.54)	6.66(0.56)	12.24(1.02)	12.42(1.04)
ASSUMES ONE-HALF OF STOPE CLOSURE RESULTS IN TROUGH SUBSIDENCE					
60-ft stope width					
		8750υε-Limiting Strain		15000υε-Limiting Strain	
N26090	2	7.61(0.63)	7.66(0.64)	13.38(1.12)	13.44(1.12)
N26930	3	7.42(0.62)	7.52(0.63)	13.19(1.10)	13.28(1.11)
N27250	4	7.41(0.62)	7.50(0.62)	13.18(1.10)	13.28(1.11)
130-foot stope width					
		8750υε-Limiting Strain		15000υε-Limiting Strain	
N26090	2	7.32(0.61)	7.34(0.61)	13.20(1.10)	13.13(1.11)
N26930	3	7.07(0.59)	7.12(0.59)	12.94(1.08)	13.09(1.09)
N27250	4	7.14(0.60)	7.24(0.60)	12.95(1.08)	13.13(1.09)

intermittent presence of water at the surface. The maximum trough subsidence induced tensile strains decrease downward from the ground surface and are relatively small even at the surface. The hillside slope over the upper stopes will not permit the impoundment of water at the surface. Water will be available to enter the slightly opened jointing only during periods of rain and during snowmelt.

The minor additional quantity of water entering the mine during operation will have to be treated as required. Water entering the upper stopes after closure of the mine and construction of bulkhead seals will be impounded in the mine pool. The eventual equilibrium condition will involve minor inflow into the upper stopes and outflow through the tight natural fracture system.

Rock Characterization, Sub-county Road Backfill Access Vicinity

During the planning stages for the Drop Pass Backfill Access, a decision was made to test the rock mass quality and hydrologic nature of the site. During March 1994, a 500-foot core hole, LFC-432, was drilled northwesterly from the 2715 West Crosscut. This hole followed the planned trace of the proposed crosscut. Results of the drilling indicated generally fragmented rock quality in clastic rocks encountered in the last half of the hole. Initial water discharge from the hole was approximately 6.5 gpm; however, the discharge decreased to about 3.3 gpm after 24 hours. At 10 days the discharge stabilized at about 2 gpm. The hole was abandoned and packed with a valved pipe.

Concern over the poor rock quality prompted the selection of another site, approximately 200 feet further north, for the supply raises. Three core holes; LFGT-1, LFGT-2 and LFGT-3; were drilled to a depth of approximately 230 feet below surface at the new location. Results of the drilling again indicated generally poor rock qualities throughout all the holes.

Overall rock mass quality from the four drill holes was low enough to cause a reevaluation of the overall backfill system. A decision was made to redesign the backfill system to take advantage of better rock conditions that exist to the east of the valley bottom. The result of this reevaluation is the Sub-county Road Backfill Access Alternative.

Further study of data from the recent drill programs, exploration drill hole logs and surface and underground mapping suggests many reasons for the poor rock mass quality in Wolfe Camp valley. The following is a geologic interpretation of those data.

Stratigraphy and Mineralogy

The clastic rocks in the vicinity of the valley bottom and to the west are probably the same stratigraphic unit as those in the mine area. They consist of thin, to very thin beds, of siltstone and sandstone, with local lenses of conglomerate. The sediments have undergone some degree of metamorphism and, therefore, react to high stress with largely brittle deformation, creating open fractures. The clastics are easily parted along bedding planes. Although the same is true for the clastic rocks near the planned stoping areas, those rocks have been mineralized with silica and most fractures and bedding planes have been "healed" by secondary mineral deposition. Very little mineralization is found as fracture filling in the clastics west of the mine. Only occasional "crusts" of pyrite and iron oxides are evident.

There are abundant exposures of magmatic intrusive rocks in the area west of Wolfe Camp valley. These intrusives are probably late Tertiary in age and are commonly rhyodacite with varying degrees of alteration. The intrusives are generally more competent than the clastics but are well jointed and fractured.

Structure

Rocks to the west of Wolfe Camp valley are significantly more disrupted by strong faulting than in the mine area. Two major sets of faulting are noted. A N-S striking, steeply west dipping set of normal faults is cut by a NE striking, moderately NW dipping set of reverse faults. The intrusive rocks appear to be injected along both sets and in some cases were offset by later fault movement. The net effect of this intense faulting was to "shatter" the brittle clastic rocks. This zone of disruption appears to become pronounced immediately west of a high-angle, normal fault that is interpreted to occur in the valley bottom.

Weathering

All of the core in the four holes drilled to test the zone showed evidence of weathering. Most fracture coatings are the oxidized product of chemical breakdown of sulfides. A deep weathering zone, possibly 200-300 feet deep, may exist below Wolfe Camp valley. Weathering out of the sulfide fillings appears to have contributed to the "shaley" nature of the surface outcrop on the west side of the valley. Some erosion of clay-rich beds may also have occurred.

REFERENCES

Abel, Jr., J.F. & F.T. Lee, 1980, Subsidence potential in shale and crystalline rocks: USGS Open-file Report No. 80-1072, 49 p

Abel, Jr., J.F. & F.T. Lee, 1984, Lithologic controls on subsidence: Trans SME/AIME, v 274, p 2028-2034

Abel, Jr., J.F., 1988, Soft Rock Pillars: Int'l. Jour. of Mining & Geol. Engrg. v. 6, p. 215-248

Babcock, C.O. & V.E. Hooker, 1977, Results of research to develop guidelines for mining near surface and underground bodies of water: U.S. Bureau of Mines Info Circ 8741, 17 p

Barton, N., R. Lien & J. Lunde, 1974, Engineering classification of rock masses for the design of tunnel support: in Rock Mechanics, v 6, n 4, p 189-236

Barton, N., R. Lien & J. Lunde, 1977, Estimation of support requirements for underground construction: in 16th U.S. Symp on Rock Mechanics, ASCE, p 163-177

Barton, N., F. Loset, R. Lien & J. Lunde, 1980, Application of the Q-system in design decisions concerning dimensions and appropriate support for underground installations: Proc 21st U.S. Symp on Rock Mechanics, p 553-562

Barton, N., R. Lien & J. Lunde, 1982, Rock reinforcement: in Underground Mining Methods Handbook, ed W.A. Hustrulid, SME, p 1540-1555

Barton, N., 1983, Application of Q-system and deformability of rock masses: Proc Intl Symp on Engrg Geol of Underground Construction, Balkema, p 51-70

Bieniawski, Z.T., 1973, Engineering classification of jointed rock masses: The Civil Engineer in S. Africa, Trans S. African Inst Civil Engrs, p 335-344

Bieniawski, Z.T., 1976, Rock mass classifications in rock engineering: in Proc Symp on Exploration for Rock Engrg, ed Z.T. Bieniawski, p 97-106

Bieniawski, Z.T., 1983, Rock mechanics design in mining and tunneling: Balkema, 454 p

Farmer, I., 1968, Engineering behavior of rocks: 2nd Ed., Chapman & Hall, 208 p

Kendorski, F.S., I. Khosla & M.M. Singh, 1979, Criteria for determining when a body of surface water constitutes a hazard to mining: Dept of Commerce NTIS PB81-194938, 364 p

McNearny, R.L. & J.F. Abel, Jr., 1993, Large-scale two-dimensional block caving model tests: Int'l Jour of Rock Mechanics & Mining Sci., v. 30, no., p. 93-110

Moore, R.C. & M.A. Nawrocki, 1980, Effects of subsidence from thick seam coal mining on hydrology: Dept of Commerce NTIS PB80-219280, 242 p

National Coal Board (British), 1975, Subsidence engineers' handbook: NCB Mining Dept, 111 p

Piggott, R.J. & P. Eynon, 1977, Ground movements from the presence of abandoned mine workings: Proc Conf on Large Ground Movements and Structures, ed J.D. Geddes, J Wiley, p 749-780

Roberts, D. & A. Moss, 1993, An evaluation of the proposed surface crown pillar design for the Lamefoot Project: Golder Associates Ltd., 13 p, 3 figs, Apr

Roberts, D. & A. Moss, 1993, Addendum 1, Surface monitoring: in An evaluation of the proposed surface crown pillar design for the Lamefoot Project, 1 p, 1 fig, Jun

Roberts, D. & A. Moss, 1993, Addendum 2: in An evaluation of the proposed surface crown pillar design for the Lamefoot Project, 5 p, 4 fig, Jun

Voight, B. & W. Pariseau, 1970, State of the predictive art in Subsidence engineering: Jour Soil Mechanics & Foundation Div, ASCE, v 96, n 3, SM2, p 721-750

Wardell, K. & Partners, 1976, Guidelines for mining under surface water: Dept of Commerce NTIS PB-264 729, 59 p

Zipf, Jr., R.K., 1992, MULSIM/NL theoretical and programmer's manual: USBM Info Circular 9321, 40 p

Zipf, Jr., R.K., 1992, MULSIM/NL application and practitioner's manual: USBM Info Circular 9322, 45 p

APPENDIX F
RESULTS OF MARCH 1993 NOISE SURVEY

RESULTS OF MARCH 1993 NOISE SURVEY AT THE LAMEFOOT PROJECT

DATE: **3-16-93** **FIRST NOISE SURVEY**
Morning Survey

Sunny, 35 degrees, light and variable breezes

Checked audio dosimeter calibration: Read 93.6 dB for a calibration level of 94.0 dB

Station #1

Start 9:10 am

Predominant sounds from highway traffic, slight from Lamefoot site.

9:11	pickup truck	76.5 dB
9:11:50	strong wind gust	62 dB
9:12:05	VW bug	91.4 dB
9:18	2 pickup trucks	78.6 & 77.0 dB
9:19:30	slight loader noise from site	48 - 51 dB
9:20	van	79.2 dB
9:21	car	76.4 dB
9:22:30	pickup truck	77.9 dB
9:23 - 9:24	loader goes up ramp	52 -55 dB
9:24:30	car	77.2 dB
9:24 - 9:25	loader on upper waste stockpile	48 - 53 dB

Stop 9:25 am

Maximum Peak: 91.4 dB
15-min. Avg : 47 dB

Station # 2

Start 8:40 am

Audible crusher noise throughout station survey (48 -56 dB).

8:40:30	car	74 dB
8:41:30	pickup truck	84 dB
8:42:20	pickup truck	78 dB
8:42:30	can	77.7 dB
8:43	car	72.3 dB
8:44	loader moves down ramp to lower waste rock stockp.	55 - 62 dB
8:45	car	73.3 dB
8:46:15	van	72.6 dB
8:46:45	pickup truck	73.5 dB

8:47	loader moves up ramp from lower waste rock stockp.	55 -60 dB
8:50	car & van	74.6 & 73.7 dB
8:53	2 pickup trucks	72.4 & 72.6 dB
8:54:30	pickup truck	74.8 dB

Stop 8:55 am

Maximum Peak: 84.2 dB
15-min. Avg : 41 dB

Station # 3

Start 9:40 am

No noise heard from the Lamefoot site.

9:41:30	2 pickup trucks	71.7 dB
9:42	Pickup truck	77 dB
9:44	18-wheeler truck	87.4 dB
9:44:30	Strong wind gust	71 dB
9:44:45	2 cars	73 & 74.5 dB
9:46	Pickup truck	72.1 dB
9:47	Van	69.6 dB
9:48	pickup truck	71.7 dB
9:50	strong wind gust	58-63 dB
9:51	pickup truck	72.7 dB
9:52	car	69.1 dB
9:52:30	pickup truck	70.2 dB
9:54	car	69.3 dB
9:55	car	69.9 dB

Stop 9:55 am

Maximum Peak: 87.4 dB
15-min. Avg : <40 dB

Station # 4

Start 10:25 am

No sound from site. Slight sound from vehicles on Highway 21 below. Vehicle noise barely registers on noise meter. Highest recorded sounds from birds and stronger breezes in trees.

10:28	Strong wind gust	45-57 dB

Stop 10:40 am

Maximum Peak: 57 dB
15-min. Avg : <40 dB

Station # 5

Start 11:12 am

No highway sounds, little to no breezes, some birds, no sounds from site.

No notable peaks.

Stop 11:27 am

Maximum Peak: ____
15-min. Avg : <40 dB

Station # 6

Start 8:16 am

Little activity at site. Also occasional sounds from shop and from crusher. Highest noises are from traffic on Highway 21.

	Throughout station monitoring crusher noise	50-52 db
8:18	loader moving up ramp	69.8 dB
8:21-stop	loader working on waste stockpile	49.5 dB
8:23:30	2 cars on highway	53.5 & 54 dB
8:25	backhoe down ramp to shop	72.7 dB
8:27:30	car on highway	47.1 dB
8:28:30	loader moving down ramp	64 dB
8:30-stop	loader working on lower waste rock stockpile	49-56 dB

Stop 8:31 am

Maximum Peak: 72.7 dB
15-min. Avg : 51 dB

Station # 7

Start 11:47 am

No activity in and out of portal. Slight hum/hiss from compressor building. Slight noise from vehicles on Highway 21.

Compressor noise throughout monitoring	50.6 dB

Stop 12:02 am

Maximum Peak: 61.7 dB
15-min. Avg : 50.6 dB

Station # 8

Start 12:27 am

Residential area on gravel road in town of Pollard adjacent to Curlew Lake. Minimal sounds from people activity, dogs barking, wind chimes. No sounds from Lamefoot site, but can slightly hear vehicles on Highway 21.

12:27	car slow drive-by	54.1 dB
12:27-12:37	loud squirrel chirping	63 dB

Stop 12:42 am

Maximum Peak: 63 dB
15-min. Avg : <40 dB

DATE: 3-18-93 **SECOND NOISE SURVEY**

Late Afternoon/Evening Survey
Partly Cloudy 45 degrees, light and variable breezes

Check calibration of instrument prior to monitoring: read 93.7 dB on a 94 dB calibration tone.

Station #1

Start 2:40 pm

2:41	Pickup truck	74.8 dB
2:41:30	Car	69.5 dB
2:43	Car	68.9 dB
2:44	Pickup truck	72.1 dB
	Also crusher starts and runs to end of survey	45-50 dB
2:45	2 cars	68.1 & 70.2 dB
2:46:20	Car	67.9 dB
2:47 - 2:49	Loader up ramp from shop to dump. Works on dump to end of sta.1 survey	46-53 dB
2:48	Large (box) truck	86.5 dB
2:50	Pickup truck	73.2 dB
2:51:30	Car	68.0 dB
2:54	Pickup truck	73.9 dB

Stop 2:55 pm

Maximum Peak:	86.5 dB
15-min. Avg :	42.3 dB

Station # 2

Start 3:01 pm

3:00	Crusher operating and Loader working on upper dump at start	49-53 dB
3:01:30	Car	69.9 dB
3:04	Pickup truck and van	72.7 & 70.5 dB
3:05-3:06	Loader down ramp to shop	48-58 dB
3:05:30	Car	70.3 dB
3:07	Crusher off- sound level drops from about 46 dB to 42 dB	
3:07	Pickup truck	78.4 dB
3:07:40	Car & pickup truck	67.0 & 73.5 dB
3:08	2 cars	69.1 & 67.9 dB

3:09	Car	67.2 dB
3:10:15	Pickup truck	76.4 dB
3:12	Large flat bed truck	84.4 dB
3:13:30	Van & Pickup truck	69.9 & 72.5 dB
3:15	Car	67.0 dB

Stop 3:16 pm

Maximum Peak: 85.8 dB
15-min. Avg : 43.2 dB

Station # 3

Start 3:20 pm

No noise heard from the Lamefoot site.

3:21:30	Car	65.1 dB
3:22	Pickup truck	72.1 dB
3:23	2 Cars and Van	68.0, 68.4, 72.3 dB
3:24	Car	64.9 dB
3:24:40	Car	68.4 dB
3:25	Gust of breeze	46 - 54 dB
3:26	2 cars	68.1 & 67.8 dB
3:27	Pickup truck	77.3 dB
3:27:30	Large 18-wheel truck	89.6 dB
3:29	Van	72.2 dB
3:30	Van & Pickup truck	71.9 & 73.2 dB
3:32	Pickup truck	74.0 dB
3:34	Car	68.8 dB
3:35	Delivery (box) truck	74.9 dB

Stop 3:35 pm

Maximum Peak: 89.6 dB
15-min. Avg : 40.1 dB

Station # 4

Start 4:25 pm

No sound from site. Slight sound from vehicles on Highway 21 below. Vehicle noise barely registers on noise meter. Highest recorded sounds from birds and stronger breezes in trees.

	Birds throughout monitoring	<40 - 44 dB
4:30 -4:33	Wind in trees	41 - 44.5 dB
4:38	Wind in trees	40 - 44 dB
4:40	Wind in trees	<41 - 42 dB

Stop 4:40 pm

Maximum Peak: 44.5 dB
15-min. Avg : <40 dB

Station # 5

Start 4:55 pm

No highway sounds, little to no breezes, some birds, no sounds from site.

No notable peaks.

Stop 5:10 pm

Maximum Peak:	----
15-min. Avg :	<40 dB

Station # 6

Start 3:44 pm

Little activity at site. Some activity personnel movement adjacent to the parking lot. Also occasional sounds from shop. Highest noises are from traffic on Highway 21.

3:44:30	Car leaves parking lot	65.1 dB
3:45	Pickup truck on highway	66.0 dB
3:46	Haul truck down ramp (passed adjacent to station)	76.4 dB
3:48	Car on highway	60.1 dB
3:49	Shop sounds	53.7 dB
3:50:30	People talking outside of office	44 - 51 dB
3:51	2 cars on highway	61.1 & 66.8 dB
3:52	Pickup truck on highway	62.3 dB
3:53:30	Car on highway	59.5 dB
3:55	Loader down ramp to south waste stockpile	76.7 dB
3:57	2 vehicles on highway	62.9 & 62.4 dB

Stop 3:59 pm

Maximum Peak:	76.7 dB
15-min. Avg :	44.6 dB

Station # 7

Start 4:03 pm

Some activity in and out of portal. Slight hum/hiss from compressor building. Slight noise from vehicles on Highway 21.

4:04	Vehicle on highway	49.0 dB
4:05	Vehicle(s) on highway	52.3 dB
4:07	Loader out of portal	87.8 dB
4:08	Vehicle on highway	46.4 dB
4:10	Shop sounds - hammering	56.2 dB
4:11	Vehicle leaving parking lot	54.1 dB
4:12	Mine personnel truck enters portal	77.5 dB
4:13	Vehicle on highway	47.2 dB
4:14	Vehicles moving near shop	53.4 dB
4:17	Vehicle on highway	49.6 dB

Stop 4:18 pm

Maximum Peak:	87.8 dB
15-min. Avg :	52.7 dB

Station # 8

Start 5:27 pm

Residential area on gravel road in town of Pollard adjacent to Curlew Lake. Some sounds from people activity, dogs barking, wind chimes, cars passing slowly, starting and stopping and car doors closing. No sounds from Lamefoot site, but can slightly hear vehicles on Highway 21.

Recorded only the few higher noises.

5:30	Car stops nearby, door closes	59.6 dB
5:34	Dog barking	52 - 58 db.

Stop 5:42 pm

Maximum Peak:	59.6 dB
15-min. Avg :	40.3 dB

DATE: 3-19-93 **THIRD NOISE SURVEY**

Mid-day survey

Sunny, 50 degrees, light and variable breezes

Check calibration of instrument prior to monitoring: read 93.6 dB on a 94 dB calibration tone.

Station #1

Start 11:21:30 am

11:21:40	pickup truck	82.1 dB
11:22-11:33	crusher operating	46-62 dB
11:24	pickup truck	79.7 dB
11:25	2 cars	76 & 69.8 dB
11:26:30	pickup truck	74.9 dB
11:31	car	72.2 dB
11:31:30	2 trucks	82.6 & 78.9 dB
11:32	2 cars	72.1 & 71.1 dB
11:32:30	pickup truck	72.4 dB
11:33:15	pickup truck	79.4 dB
11:34	van	75 dB
11:35:15	car and 2 pickup trucks	75.2, 74.8 & 72.2 dB

Stop 11:37 am

Maximum Peak: 82.6 dB
15-min. Avg : 44.9 dB

Station # 2

Start 11:00 am

	Crusher working throughout monitoring	42-44.5 dB
11:00-11:03:30	loader repairing parking lot	45-49 dB
11:02:30	car	69 dB
11:03	wind gust	44.8 dB
11:05	van	72.3 dB
11:06:30	car turning	53.6 dB
11:07-stop	loader repairing parking lot	45-47 dB
11:09	car	68.4 dB
11:09:40	car	71.2 dB
11:11	truck turns onto site	58.3 dB
11:13	car	70.9 dB
11:13:30	car	71.9 dB
11:15	truck	87 dB

Stop 11:15 am

Maximum Peak: 87 dB
15-min. Avg : 49.4 dB

Station # 3

Start 2:45 pm

No noise heard from the Lamefoot site.

2:46	car	69.5 dB
2:47	flatbed truck	79.5 dB
2:48	2 cars	71.3 & 70.5 dB
2:48:30	pickup truck	74 dB
2:49:30	car	73.5 dB
2:51	pickup truck	75.1 dB
2:51:30	2 cars	68.9 & 72.2 dB
2:53	pickup truck and car	73.5 & 69.9 dB
2:55	pickup truck	74.1 dB
2:56	car	71.6 dB
2:57	van	70 dB
2:58:30	car	68.6 dB
2:59:30	car	68.9 dB

Stop 3:00 pm

Maximum Peak: 79.8 dB
15-min. Avg : 44.1 dB

Station # 4

Start 1:55 pm

No sound from site. Slight sound from vehicles on Highway 21 below. Vehicle noise barely registers on noise meter. Highest recorded sounds from birds and stronger breezes in trees.

Barely audible ventilation borehole (is line of site).

No notable peaks.

Stop 2:10 pm

Maximum Peak: --- dB
15-min. Avg : 40.5 dB

Station # 5

Start 2:20 pm

No highway sounds, little to no breezes, some birds, no sounds from site.

No notable peaks.

Stop 2:35 pm

Maximum Peak: ----
15-min. Avg : <40 dB

Station # 6

Start 10:40 am

Little activity at site. Some activity personnel movement adjacent to the parking lot. Also occasional sounds from shop. Highest noises are from traffic on Highway 21 (average highway noise approximately 48-51 dB).

10:45	airplane fly-over	61.3 dB
10:45-10:55	loader working on middle waste rock stockpile	42-45 dB
10:46	person walking in parking lot	51 dB

Stop 10:55 am

Maximum Peak: 61.3 dB
15-min. Avg : 42.8 dB

Station # 7

Start 10:17 am

No activity in and out of portal. Slight hum/hiss from compressor building. Slight noise from vehicles on Highway 21.

10:26	pickup truck into mine	65 dB

Stop 10:27

Maximum Peak: 65 dB
15-min. Avg : 51.3 dB

Station # 8

Start 3:20 pm

Some sounds from people activity, dogs barking, wind chimes, cars passing slowly. Slight loader noise from mine. Loader noise audible but below detection limit on instrument. Can slightly hear vehicles on Highway 21.

Recorded only the few higher noises.

3:29	car rolls by slowly	63.6 dB
3:33	car parks in front of house 150 feet away	60.9 dB

Stop 3:35 pm

Maximum Peak: 63.6 dB
15-min. Avg : 42.6 dB

NOISE SAMPLING OF VENTILATION BOREHOLE AT EXHAUST

Sound of ventilator is a steady wine with a low rumble.

- at ground surface just inside cage: 94.7 dB
- at top of cage 3 feet above ground surface: 90.6 db
- at ear level on outside edge of concrete pad: 91.1 dB
- 10 feet from edge of concrete pad: 68.5 dB
- 50 feet from edge of concrete pad: 58.3 dB
- 100 feet from edge of concrete pad: 47.2 dB
- 200 feet from edge of concrete pad: 43 dB
- 400 feet from edge of concrete pad: <42 dB (sound of cars on highway and wind in trees equal to 42 dB)

APPENDIX G
LAMEFOOT PROJECT WATER QUALITY ANALYTICAL DATA

APPENDIX G-1

ALLUVIAL MONITORING WELLS LF-1 AND LF-2

SVL ANALYTICAL

Client: ECHO BAY
Client Sample ID: LF-1
Sample Collection Date: 12/02/92 12:15 PM
Date of Sample Receipt: 12/07/92
Date of Report: 12/21/92

SVL Job Number: 21571
SVL Sample ID: W35183
Area:
Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	M	ANALYTE	RESULT	UNIT	DILN	M
pH	8.07	units			Molybdenum-D	< .008	mg/L		P
Alkalinity	281.	mg/L			Nickel-D	< .005	mg/L		P
Bicarbonate	278.	mg/L			Potassium-D	2.8	mg/L		P
WAD Cyanide	< .01	mg/L		C	Selenium-D	< .06	mg/L		P
Aluminum-D	.05	mg/L		P	Silver-D	< .004	mg/L		P
Arsenic-D	< .05	mg/L		P	Sodium-D	13.5	mg/L		P
Barium-D	.035	mg/L		P	Zinc-D	.003	mg/L		P
Boron-D	< .005	mg/L		P	Chloride	2.6	mg/L		I
Cadmium-D	< .004	mg/L		P	Fluoride	.3	mg/L		I
Calcium-D	145.	mg/L		P	Sulfate	226.	mg/L		I
Chromium-D	< .003	mg/L		P	NO3+NO2-N	.69	mg/L		C
Copper-D	< .004	mg/L		P	o-Phosphate	.60	mg/L		I
Iron-D	.056	mg/L		P	Conductance	897.	mg/L		
Lead-D	< .05	mg/L		P	TDS	572.	mg/L		
Magnesium-D	37.4	mg/L		P	TSS	310.	mg/L		
Manganese-D	.006	mg/L		P	Oil & Grease	<1.0	mg/L		
Mercury-D	< .0002	mg/L							

M(ethod): P=ICP F=GFAA V=CVAA C=COLORIMETRIC I=ION CHROMATOGRAPHY

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

DATE 12/22/92

Client: ECHO BAY
Client Sample ID: LF-2
Sample Collection Date: 12/02/92 NOON
Date of Sample Receipt: 12/07/92
Date of Report: 12/21/92

SVL Job Number: 21571
SVL Sample ID: W35184
Area:
Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	M	ANALYTE	RESULT	UNIT	DILN	M
pH	7.80	units			Molybdenum-D	< .008	mg/L		P
Alkalinity	301.	mg/L			Nickel-D	< .005	mg/L		P
Bicarbonate	299.	mg/L			Potassium-D	2.5	mg/L		P
WAD Cyanide	< .01	mg/L		C	Selenium-D	< .06	mg/L		P
Aluminum-D	.06	mg/L		P	Silver-D	< .004	mg/L		P
Arsenic-D	< .05	mg/L		P	Sodium-D	14.0	mg/L		P
Barium-D	.034	mg/L		P	Zinc-D	.009	mg/L		P
Boron-D	< .005	mg/L		P	Chloride	5.6	mg/L		I
Cadmium-D	< .004	mg/L		P	Fluoride	.3	mg/L		I
Calcium-D	176.	mg/L		P	Sulfate	287.	mg/L		I
Chromium-D	< .003	mg/L		P	NO3+NO2-N	.49	mg/L		C
Copper-D	< .004	mg/L		P	o-Phosphate	5.60	mg/L		I
Iron-D	.083	mg/L		P	Conductance	1010	mg/L		
Lead-D	< .05	mg/L		P	TDS	875.	mg/L		
Magnesium-D	42.2	mg/L		P	TSS	1830	mg/L		
Manganese-D	.005	mg/L		P	Oil & Grease	<1.0	mg/L		
Mercury-D	< .0002	mg/L							

M(ethod): P=ICP F=GFAA V=CVAA C=COLORIMETRIC I=ION CHROMATOGRAPHY

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

DATE 12/22/92

SVL ANALYTICAL

Client: ECHO BAY
Client Sample ID: LF-2
Sample Collection Date: 1/05/93 11:00
Date of Sample Receipt: 01/13/93
Date of Report: 01/25/93

SVL Job Number: 30029
SVL Sample ID: W36253
Area:
Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	M	ANALYTE	RESULT	UNIT	DILN	M
pH	7.70	units			Potassium-D	2.8	mg/L		P
Alkalinity	303.	mg/L			Selenium-D	< .04	mg/L		P
Bicarbonate	303.	mg/L			Sodium-D	13.3	mg/L		P
WAD Cyanide	< .01	mg/L		C	Zinc-D	.012	mg/L		P
Aluminum-D	.09	mg/L		P	Chloride	6.63	mg/L		I
Arsenic-D	< .04	mg/L		P	Fluoride	.24	mg/L		I
Boron-D	.01	mg/L		P	Sulfate	297.	mg/L		I
Cadmium-D	< .002	mg/L		P	NO3+NO2-N	1.03	mg/L		C
Calcium-D	169.	mg/L		P	o-Phosphate	4.69	mg/L		I
Copper-D	.004	mg/L		P	Conductance	1030	umhos		
Iron-D	.088	mg/L		P	TDS	686.	mg/L		
Lead-D	< .03	mg/L		P	TSS	1950	mg/L		
Magnesium-D	40.0	mg/L		P	Cyanide	< .01	mg/L		C
Manganese-D	< .002	mg/L		P	Barium -D	.030	mg/L		P
Mercury-D	< .0002	mg/L		V	Chromium -D	< .006	mg/L		P
Molybdenum-	< .007	mg/L		P	Silver -D	< .004	mg/L		P
Nickel-D	< .01	mg/L		P					

C/A BAL: 12.6 %diff CATION SUM: 12.4 meq/L ANION SUM: 9.6 meq/L

M(ethods): P=ICP F=GFAA V=CVAA C=COLORIMETRIC I=ION CHROMATOGRAPHY L=LECO

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

Khelha Bencich by dc DATE 01/25/93

SVL ANALYTICAL

Client: ECHO BAY
Client Sample ID: LF-1
Sample Collection Date: 1/05/93 10:45
Date of Sample Receipt: 01/13/93
Date of Report: 01/25/93

SVL Job Number: 30029
SVL Sample ID: W36252
Area:
Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	M	ANALYTE	RESULT	UNIT	DILN	M
pH	8.00	units			Potassium-D	2.7	mg/L		P
Alkalinity	282.	mg/L			Selenium-D	< .04	mg/L		P
Bicarbonate	282.	mg/L			Sodium-D	11.1	mg/L		P
WAD Cyanide	< .01	mg/L		C	Zinc-D	.009	mg/L		P
Aluminum-D	.11	mg/L		P	Chloride	2.98	mg/L		I
Arsenic-D	.04	mg/L		P	Fluoride	.25	mg/L		I
Boron-D	.01	mg/L		P	Sulfate	218.	mg/L		I
Cadmium-D	< .002	mg/L		P	NO3+NO2-N	1.99	mg/L		C
Calcium-D	137.	mg/L		P	o-Phosphate	1.02	mg/L		I
Copper-D	.004	mg/L		P	Conductance	862.	umhos		
Iron-D	.124	mg/L		P	TDS	571.	mg/L		
Lead-D	.08	mg/L		P	TSS	453.	mg/L		
Magnesium-D	35.6	mg/L		P	Cyanide	< .01	mg/L		C
Manganese-D	< .002	mg/L		P	Barium-D	.031	mg/L		P
Mercury-D	< .0002	mg/L		V	Chromium-D	< .006	mg/L		P
Molybdenum-	< .007	mg/L		P	Silver-D	< .004	mg/L		P
Nickel-D	< .01	mg/L		P					

C/A BAL: 15.2 %diff CATION SUM: 10.3 meq/L ANION SUM: 7.6 meq/L

M(ethods): P=ICP F=GFAA V=CVAA C=COLORIMETRIC I=ION CHROMATOGRAPHY L=LECO

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

DATE 01/25/93

SVL ANALYTICAL

Client: ECHO BAY
Client Sample ID: LF1
Sample Collection Date: 2/03/93 10:45A
Date of Sample Receipt: 02/03/93
Date of Report: 02/19/93

SVL Job Number: 30094
SVL Sample ID: W36830

Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	M	ANALYTE	RESULT	UNIT	DILN	M
pH	7.80	units			Mercury-D	< .0002	mg/L		V
Alkalinity	271.	mg/L			Molybdenum-D	< .007	mg/L		P
Bicarbonate	271.	mg/L			Nickel-D	< .01	mg/L		P
WAD Cyanide	< .01	mg/L		C	Potassium-D	2.2	mg/L		P
Aluminum-D	< .03	mg/L		P	Selenium-D	< .04	mg/L		P
Arsenic-D	< .04	mg/L		P	Sodium-D	11.5	mg/L		P
Boron-D	< .01	mg/L		P	Zinc-D	.021	mg/L		P
Cadmium-D	< .002	mg/L		P	Chloride	3.23	mg/L		I
Calcium-D	140.	mg/L		P	Fluoride	.20	mg/L		I
Copper-D	< .003	mg/L		P	Sulfate	221.	mg/L		I
Iron-D	< .009	mg/L		P	NO3+NO2-N	.58	mg/L		C
Lead-D	< .03	mg/L		P	Conductance	851.	umhos		
Magnesium-D	37.8	mg/L		P	TDS	619.	mg/L		
Manganese-D	< .002	mg/L		P	TSS	357.	mg/L		
Ortho-Phosphate	<.01	mg/L			Silver-D	< .005	mg/L		P
Barium-D	.03	mg/L		P	Chromium-D	< .006	mg/L		P

M(ethods): P=ICP F=GFAA V=CVAA C=COLORIMETRIC I=ION CHROMATOGRAPHY L=LECO

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

D. Brick DATE 02/19/93

SVL ANALYTICAL

lient: ECHO BAY - WALT HUNT
l[illegible]t Sample ID: LF-1
ample Collection Date: 3/02/93 9:40
ate of Sample Receipt: 03/04/93
Date of Report: 03/16/93

SVL Job Number: 30163
SVL Sample ID: W37685
Area:
Matrix: WATER

ALYTE	RESULT	UNIT	DILN	M	ANALYTE	RESULT	UNIT	DILN	M
	7.84	units		W	Molybdenum-D	< .007	mg/L		P
alinity	270.	mg/L		T	Nickel-D	< .01	mg/L		P
arbonate	270.	mg/L		T	Potassium-D	1.5	mg/L		P
Cyanide	< .01	mg/L		C	Selenium-D	< .04	mg/L		P
minum-D	< .03	mg/L		P	Sodium-D	11.6	mg/L		P
enic-D	< .04	mg/L		P	Zinc-D	.015	mg/L		P
n-D	< .01	mg/L		P	Chloride	2.27	mg/L		I
ium-D	< .002	mg/L		P	Fluoride	.26	mg/L		I
ium-D	130.	mg/L		P	Sulfate	204.	mg/L	20	I
er-D	< .003	mg/L		P	NO3+NO2-N	.93	mg/L		C
D	< .009	mg/L		P	o-Phosphate	.21	mg/L		I
D	< .03	mg/L		P	Conductance	810.	umhos		
sium-D	35.5	mg/L		P	TDS	578.	mg/L		W
nese-D	< .002	mg/L		P	TSS	327.	mg/L		W
y-D	< .0002	mg/L		P					

L: 17.0 %diff CATION SUM: 10.0 meq/L ANION SUM: 7.1 meq/L

[illegible] F=GFAA V=CVAA C=COLORIMETRIC I=ION CHROMATOGRAPHY L=LECO T=TITRATION W=WET CHEM E=ELECTRODE

PORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

DATE 03/16/93

SVL ANALYTICAL

Client: ECHO BAY
Client Sample ID: LF2
Sample Collection Date: 2/03/93 11:20A
Date of Sample Receipt: 02/03/93
Date of Report: 02/19/93

SVL Job Number: 30094
SVL Sample ID: W36831

Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	M	ANALYTE	RESULT	UNIT	DILN	M
pH	7.70	units			Mercury-D	< .0002	mg/L		V
Alkalinity	300.	mg/L			Molybdenum- D	< .007	mg/L		P
Bicarbonate	300.	mg/L			Nickel-D	< .01	mg/L		P
WAD Cyanide	< .01	mg/L		C	Potassium-D	2.8	mg/L		P
Aluminum-D	< .03	mg/L		P	Selenium-D	< .04	mg/L		P
Arsenic-D	< .04	mg/L		P	Sodium-D	13.6	mg/L		P
Boron-D	< .01	mg/L		P	Zinc-D	.007	mg/L		P
Cadmium-D	< .002	mg/L		P	Chloride	6.28	mg/L		I
Calcium-D	173.	mg/L		P	Fluoride	.18	mg/L		I
Copper-D	< .003	mg/L		P	Sulfate	289.	mg/L		I
Iron-D	< .009	mg/L		P	NO3+NO2-N	.43	mg/L		C
Lead-D	< .03	mg/L		P	Conductance	1010	umhos		
Magnesium-D	42.6	mg/L		P	TDS	728.	mg/L		
Manganese-D	< .002	mg/L		P	TSS	3110	mg/L		
Ortho-Phosphate	<.01	mg/L			Silver-D	< .005	mg/L		P
Barium-D	.03	mg/L		P	Chromium-D	< .006	mg/L		P

M(ethods): P=ICP F=GFAA V=CVAA C=COLORIMETRIC I=ION CHROMATOGRAPHY L=LECO

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

DATE 02/19/93

SVL ANALYTICAL

Client: ECHO BAY - WALT HUNT
Client Sample ID: LF-2
Sample Collection Date: 3/02/93 10:10
ate of Sample Receipt: 03/04/93
Date of Report: 03/16/93

SVL Job Number: 30163
SVL Sample ID: W37686
Area:
Matrix: WATER

NALYTE	RESULT	UNIT	DILN	M	ANALYTE	RESULT	UNIT	DILN	M
H	7.78	units		W	Molybdenum-D	< .007	mg/L		P
lkalinity	302.	mg/L		T	Nickel-D	< .01	mg/L		P
icarbonate	302.	mg/L		T	Potassium-D	2.9	mg/L		P
AD Cyanide	< .01	mg/L		C	Selenium-D	< .04	mg/L		P
uminum-D	< .03	mg/L		P	Sodium-D	13.6	mg/L		P
senic-D	< .04	mg/L		P	Zinc-D	.011	mg/L		P
ron-D	< .01	mg/L		P	Chloride	6.08	mg/L		I
dmium-D	< .002	mg/L		P	Fluoride	.28	mg/L		I
lcium-D	175.	mg/L		P	Sulfate	321.	mg/L	20	I
pper-D	.004	mg/L		P	NO3+NO2-N	.68	mg/L		C
on-D	< .009	mg/L		P	o-Phosphate	1.84	mg/L		I
d-D	< .03	mg/L		P	Conductance	1028.	umhos		
nesium-D	42.5	mg/L		P	TDS	754.	mg/L		W
ganese-D	< .002	mg/L		P	TSS	1560	mg/L		W
cury-D	< .0002	mg/L		P					

BAL: 12.8 %diff CATION SUM: 12.9 meq/L ANION SUM: 10.0 meq/L

s): P=ICP F=GFAA V=CVAA C=COLORIMETRIC I=ION CHROMATOGRAPHY L=LECO T=TITRATION W=WET CHEM E=ELECTRODE

REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

DATE 3/16/93

SVL ANALYTICAL

Client: ECHO BAY
Client Sample ID: LF-1
Sample Collection Date: 4/06/93 NOON
Date of Sample Receipt: 04/09/93
Date of Report: 04/23/93

SVL Job Number: 30299
SVL Sample ID: W38807
Area:
Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	M	ANALYTE	RESULT	UNIT	DILN	M
pH	7.90	units		W	Molybdenum-D	< .008	mg/L		P
Alkalinity	256.	mg/L		T	Nickel-D	< .01	mg/L		P
Bicarbonate	256.	mg/L		T	Potassium-D	2.6	mg/L		P
WAD Cyanide	< .01	mg/L		C	Selenium-D	.04	mg/L		P
Aluminum-D	.03	mg/L		P	Sodium-D	11.4	mg/L		P
Arsenic-D	< .04	mg/L		P	Zinc-D	.003	mg/L		P
Boron-D	< .006	mg/L		P	Chloride	2.02	mg/L		I
Cadmium-D	< .002	mg/L		P	Fluoride	.21	mg/L		I
Calcium-D	116.	mg/L		P	Sulfate	154.	mg/L		I
Copper-D	< .004	mg/L		P	NO3+NO2-N	1.28	mg/L		C
Iron-D	.018	mg/L		P	o-Phosphate	.07	mg/L		I
Lead-D	< .04	mg/L		P	Conductance	717.	umhos		
Magnesium-D	31.4	mg/L		P	TDS	474.	mg/L		W
Manganese-D	< .001	mg/L		P	TSS	111.	mg/L		W
Mercury-D	< .0002	mg/L		V					

M(ethods): P=ICP F=GFAA V=CVAA C=COLORIMETRIC I=ION CHROMATOGRAPHY L=LECO T=TITRATION W=WET CHEM E=ELECTRODE

CONDUCTIVITY RESULT WAS ADJUSTED TO 25 DEGREES C

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

DATE 04/26/93

SVL ANALYTICAL

Client: ECHO BAY
Client Sample ID: LF-2
Sample Collection Date: 4/06/93 11:15 AM
Date of Sample Receipt: 04/09/93
Date of Report: 04/23/93

SVL Job Number: 30299
SVL Sample ID: W38808
Area:
Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	M	ANALYTE	RESULT	UNIT	DILN	M
pH	7.96	units		W	Molybdenum-D	< .008	mg/L		P
Alkalinity	297.	mg/L		T	Nickel-D	< .01	mg/L		P
Bicarbonate	297.	mg/L		T	Potassium-D	2.8	mg/L		P
WAD Cyanide	< .01	mg/L		C	Selenium-D	< .04	mg/L		P
Aluminum-D	< .02	mg/L		P	Sodium-D	12.1	mg/L		P
Arsenic-D	< .04	mg/L		P	Zinc-D	.013	mg/L		P
Boron-D	< .006	mg/L		P	Chloride	6.54	mg/L		I
Cadmium-D	< .002	mg/L		P	Fluoride	.19	mg/L		I
Calcium-D	174.	mg/L		P	Sulfate	327.	mg/L		I
Copper-D	< .004	mg/L		P	NO3+NO2-N	.92	mg/L		C
Iron-D	.022	mg/L		P	o-Phosphate	.01	mg/L		I
Lead-D	< .04	mg/L		P	Conductance	1050	umhos		
Magnesium-D	49.3	mg/L		P	TDS	782.	mg/L		W
Manganese-D	< .001	mg/L		P	TSS	23.4	mg/L		W
Mercury-D	< .0002	mg/L		V					

M(ethods): P=ICP F=GFAA V=CVAA C=COLORIMETRIC I=ION CHROMATOGRAPHY L=LECO T=TITRATION W=WET CHEM E=ELECTRODE

CONDUCTIVITY RESULT WAS ADJUSTED TO 25 DEGREES C

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

DATE 04/26/93

SVL ANALYTICAL

Formerly Silver Valley Laboratories, Inc.

SVL Job Number: 30449
SVL Sample ID: W39861

Client: ECHO BAY
Collection Date: 05/11/93 11:45 AM
Date of Receipt: 05/14/93
Date of Report: 06/08/93

Sample: LF-1

Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	M	ANALYTE	RESULT	UNIT	DILN	M
pH	7.72	units		W	Molybdenum-D	<0.008	mg/L		P
Alkalinity	240	mg/L		T	Nickel-D	<0.015	mg/L		P
Bicarbonate	240	mg/L		T	Potassium-D	2.6	mg/L		P
Cyanide	<0.01	mg/L		C	Selenium-D	<0.04	mg/L		P
Aluminum-D	<0.02	mg/L		P	Sodium-D	11.3	mg/L		P
Arsenic-D	<0.04	mg/L		P	Zinc-D	0.003	mg/L		P
Boron-D	<0.006	mg/L		P	Chloride	3.51	mg/L		I
Cadmium-D	0.002	mg/L		P	Fluoride	0.22	mg/L		I
Calcium-D	118	mg/L		P	Sulfate	181	mg/L		I
Copper-D	<0.004	mg/L		P	NO3+NO2-N	1.27	mg/L		C
Iron-D	<0.009	mg/L		P	o-Phosphate	0.06	mg/L		I
Lead-D	<0.04	mg/L		P	Conductance	765	umhos		
Magnesium-D	32.2	mg/L		P	TDS	455	mg/L		W
nganese-D	<0.001	mg/L		P	TSS	58.6	mg/L		W
rcury-D	<0.0002	mg/L		V	Silver-D	<0.002	mg/L		P
Barium-D	0.026	mg/L		P	Chromium-D	0.004	mg/L		P

M(ethods): P=200.7 F=GFAA V=245.2 C=COLORIMETRIC I=300.0 L=EPA600/2-78-054 T=TITRATION W=WET CHEM E=ELECTRODE

This report has been reviewed and is certified to be accurate:

Joe Costello / [illegible]
Joe Costello, Project Manager DATE 06/08/93

– An Inc. 500 Company –

SVL ANALYTICAL

Formerly Silver Valley Laboratories, Inc.

SVL Job Number: 30449
SVL Sample ID: W39862

Client: ECHO BAY
Collection Date: 05/11/93 11:30 AM
Date of Receipt: 05/14/93
Date of Report: 06/08/93

Sample: LF-2

Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	M	ANALYTE	RESULT	UNIT	DILN	M
pH	7.70	units		W	Molybdenum-D	<0.008	mg/L		P
Alkalinity	250	mg/L		T	Nickel-D	<0.015	mg/L		P
Bicarbonate	250	mg/L		T	Potassium-D	2.8	mg/L		P
Cyanide	<0.01	mg/L		C	Selenium-D	<0.04	mg/L		P
Aluminum-D	<0.02	mg/L		P	Sodium-D	10.9	mg/L		P
Arsenic-D	<0.04	mg/L		P	Zinc-D	0.006	mg/L		P
Boron-D	<0.006	mg/L		P	Chloride	4.32	mg/L		I
Cadmium-D	<0.002	mg/L		P	Fluoride	0.20	mg/L		I
Calcium-D	126	mg/L		P	Sulfate	227	mg/L		I
Copper-D	<0.004	mg/L		P	NO3+NO2-N	0.76	mg/L		C
Iron-D	<0.009	mg/L		P	o-Phosphate	0.07	mg/L		I
Lead-D	<0.04	mg/L		P	Conductance	848	umhos		
Magnesium-D	37.4	mg/L		P	TDS	513	mg/L		W
Manganese-D	<0.001	mg/L		P	TSS	73.3	mg/L		W
Mercury-D	<0.0002	mg/L		V	Silver-D	<0.002	mg/L		P
Barium-D	0.026	mg/L		P	Chromium-D	0.004	mg/L		P

M(ethods): P=200.7 F=GFAA V=245.2 C=COLORIMETRIC I=300.0 L=EPA600/2-78-054 T=TITRATION W=WET CHEM E=ELECTRODE

This report has been reviewed and is certified to be accurate:

Joe Costello, Project Manager DATE 06/08/93

SVL ANALYTICAL

Client: ECHO BAY MINERALS, INC.
Client Sample ID: LF-1
Sample Collection Date: 04/07/92
Date of Sample Receipt: 04/14/92
Date of Report: 5/ 1/92

SVL Job Number: 20387
SVL Sample ID: W23621
Time: 1:30 PM
Matrix: WATER

ANALYTE		RESULT	UNIT	DILN
pH		7.72	units	
Alkalinity		254.	mg/L	
Bicarbonate		254.	mg/L	
WAD Cyanide	<	.01	mg/L	
Aluminum-D	<	.04	mg/L	
Arsenic-D		.026	mg/L	
Boron-D	<	.02	mg/L	
Cadmium-D	<	.003	mg/L	
Calcium-D		128.10	mg/L	
Copper-D	<	.005	mg/L	
Iron-D		.04	mg/L	
Lead-D	<	.001	mg/L	
Magnesium-D		34.25	mg/L	
Manganese-D	<	.001	mg/L	
Mercury-D	<	.0002	mg/L	

ANALYTE		RESULT	UNIT	DILN
Molybdenum-D	<	.02	mg/L	
Nickel-D	<	.02	mg/L	
Potassium-D		2.	mg/L	
Selenium-D		.012	mg/L	
Sodium-D		9.66	mg/L	
Zinc-D		.006	mg/L	
Chloride		1.9	mg/L	
Fluoride		.3	mg/L	
Sulfate		203.0	mg/L	
NO3+NO2-N		.59	mg/L	
o-Phosphate		1.14	mg/L	
Conductance		806.	umhos	
TDS		552.	mg/L	
TSS		863.0	mg/L	
Oil & Grease		--NA--		

--NA-- NOT ANALYZED

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

M. Bencich by SH DATE 05/04/92

SVL ANALYTICAL

Client: ECHO BAY MINERALS, INC.
Client Sample ID: LF-2
Sample Collection Date: 04/07/92
Date of Sample Receipt: 04/14/92
Date of Report: 5/ 1/92

SVL Job Number: 20387
SVL Sample ID: W23622
Time: 2 PM
Matrix: WATER

ANALYTE	RESULT	UNIT	DILN
pH	7.89	units	
Alkalinity	292.	mg/L	
Bicarbonate	292.	mg/L	
WAD Cyanide	< .01	mg/L	
Aluminum-D	< .04	mg/L	
Arsenic-D	< .002	mg/L	
Boron-D	< .02	mg/L	
Cadmium-D	< .003	mg/L	
Calcium-D	174.10	mg/L	
Copper-D	.009	mg/L	
Iron-D	.04	mg/L	
Lead-D	< .001	mg/L	
Magnesium-D	49.08	mg/L	
Manganese-D	.002	mg/L	
Mercury-D	< .0002	mg/L	

ANALYTE	RESULT	UNIT	DILN
Molybdenum-D	< .02	mg/L	
Nickel-D	< .02	mg/L	
Potassium-D	3.	mg/L	
Selenium-D	.020	mg/L	
Sodium-D	12.40	mg/L	
Zinc-D	.025	mg/L	
Chloride	8.5	mg/L	
Fluoride	.3	mg/L	
Sulfate	335.0	mg/L	
NO3+NO2-N	.52	mg/L	
o-Phosphate	.17	mg/L	
Conductance	1078	umhos	
TDS	799.	mg/L	
TSS	103.2	mg/L	
Oil & Grease	--NA--		

--NA-- NOT ANALYZED

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

M Bencick by SH DATE 05/04/92

SVL ANALYTICAL

Client: ECHO BAY MINERALS, INC.
Client Sample ID: LF-1
Sample Collection Date: 05/05/92 9:00AM
Date of Sample Receipt: 05/07/92
Date of Report: 5/19/92

SVL Job Number: 20489
SVL Sample ID: W24376
Area:
Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	ANALYTE	RESULT	UNIT	DILN
pH	8.02	units		Mercury-D	< .0002	mg/L	
Alkalinity	247.	mg/L		Molybdenum-D	< .02	mg/L	
Bicarbonate	247.	mg/L		Nickel-D	< .02	mg/L	
WAD Cyanide	< .01	mg/L		Potassium-D	3.	mg/L	
Aluminum-D	.07	mg/L		Selenium-D	< .002	mg/L	
Arsenic-D	.047	mg/L		Silver-D	--NA--		
Barium-D	--NA--			Sodium-D	9.16	mg/L	
Boron-D	< .02	mg/L		Zinc-D	< .005	mg/L	
Cadmium-D	< .003	mg/L		Chloride	2.2	mg/L	
Calcium-D	135.60	mg/L		Fluoride	.3	mg/L	
Chromium-D	--NA--			Sulfate	214.0	mg/L	
Copper-D	< .005	mg/L		NO3+NO2-N	.99	mg/L	
Iron-D	.09	mg/L		o-Phosphate	< .05	mg/L	
Lead-D	.019	mg/L		Conductance	769.	umhos	
Magnesium-D	35.00	mg/L		TDS	544.	mg/L	
Manganese-D	.004	mg/L		TSS	50.9	mg/L	

--NA-- NOT ANALYZED

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

DATE 05/20/92

SVL ANALYTICAL

Client: ECHO BAY MINERALS, INC.
Client Sample ID: LF-2
Sample Collection Date: 05/05/92 8:45AM
Date of Sample Receipt: 05/07/92
Date of Report: 5/19/92

SVL Job Number: 20489
SVL Sample ID: W24377
Area:
Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	ANALYTE	RESULT	UNIT	DILN
pH	7.89	units		Mercury-D	< .0002	mg/L	
Alkalinity	264.	mg/L		Molybdenum-D	< .02	mg/L	
Bicarbonate	264.	mg/L		Nickel-D	< .02	mg/L	
WAD Cyanide	< .01	mg/L		Potassium-D	3.	mg/L	
Aluminum-D	.33	mg/L		Selenium-D	.003	mg/L	
Arsenic-D	.033	mg/L		Silver-D	--NA--		
Barium-D	--NA--			Sodium-D	9.21	mg/L	
Boron-D	< .02	mg/L		Zinc-D	.013	mg/L	
Cadmium-D	< .003	mg/L		Chloride	6.4	mg/L	
Calcium-D	176.90	mg/L		Fluoride	.4	mg/L	
Chromium-D	--NA--			Sulfate	319.0	mg/L	
Copper-D	.007	mg/L		NO3+NO2-N	.55	mg/L	
Iron-D	.60	mg/L		o-Phosphate	< .05	mg/L	
Lead-D	.002	mg/L		Conductance	955.	umhos	
Magnesium-D	47.44	mg/L		TDS	776.	mg/L	
Manganese-D	.021	mg/L		TSS	1658	mg/L	

--NA-- NOT ANALYZED

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

Melva Bencick by BL DATE 05/20/92

SVL ANALYTICAL

Client: ECHO BAY MINERALS, INC.
Client Sample ID: LF-1
Sample Collection Date: 06/09/92 10:30AM
Date of Sample Receipt: 06/15/92
Date of Report: 6/30/92

SVL Job Number: 20666
SVL Sample ID: W25738
Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	ANALYTE	RESULT	UNIT	DILN
pH	7.76	units		Mercury-D	< .0002	mg/L	
Alkalinity	234.	mg/L		Molybdenum-D	< .02	mg/L	
Bicarbonate	234.	mg/L		Nickel-D	< .02	mg/L	
WAD Cyanide	< .01	mg/L		Potassium-D	2.	mg/L	
Aluminum-D	.91	mg/L		Selenium-D	< .05	mg/L	
Arsenic-D	< .04	mg/L		Silver-D	< .006	mg/L	
Barium-D	.044	mg/L		Sodium-D	9.54	mg/L	
Boron-D	< .02	mg/L		Zinc-D	.021	mg/L	
Cadmium-D	.003	mg/L		Chloride	2.7	mg/L	
Calcium-D	126.10	mg/L		Fluoride	.3	mg/L	
Chromium-D	< .005	mg/L		Sulfate	211.0	mg/L	20X
Copper-D	.006	mg/L		NO3+NO2-N	.87	mg/L	
Iron-D	1.48	mg/L		o-Phosphate	.69	mg/L	
Lead-D	< .04	mg/L		Conductance	798.	umhos	
Magnesium-D	34.36	mg/L		TDS	616.	mg/L	
Manganese-D	.028	mg/L		TSS	292.5	mg/L	
				Cyanide	< .01	mg/L	

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

M. Bencich by SH DATE 06/30/92

SVL ANALYTICAL

Client: ECHO BAY MINERALS, INC.
Client Sample ID: LF-2
Sample Collection Date: 06/09/92 10:15AM
Date of Sample Receipt: 06/15/92
Date of Report: 6/30/92

SVL Job Number: 20666
SVL Sample ID: W25739
Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	ANALYTE	RESULT	UNIT	DILN
pH	7.65	units		Mercury-D	.0002	mg/L	
Alkalinity	270.	mg/L		Molybdenum-D	< .02	mg/L	
Bicarbonate	270.	mg/L		Nickel-D	< .02	mg/L	
WAD Cyanide	< .01	mg/L		Potassium-D	3.	mg/L	
Aluminum-D	.27	mg/L		Selenium-D	< .05	mg/L	
Arsenic-D	< .04	mg/L		Silver-D	< .006	mg/L	
Barium-D	.033	mg/L		Sodium-D	9.73	mg/L	
Boron-D	< .02	mg/L		Zinc-D	.010	mg/L	
Cadmium-D	< .003	mg/L		Chloride	4.7	mg/L	
Calcium-D	147.20	mg/L		Fluoride	.3	mg/L	
Chromium-D	< .005	mg/L		Sulfate	267.0	mg/L	20X
Copper-D	.006	mg/L		NO3+NO2-N	.47	mg/L	
Iron-D	.43	mg/L		o-Phosphate	5.12	mg/L	
Lead-D	< .04	mg/L		Conductance	904.	umhos	
Magnesium-D	42.01	mg/L		TDS	691.	mg/L	
Manganese-D	.006	mg/L		TSS	1750	mg/L	
				Cyanide	< .01	mg/L	

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

M. Bencick by SH DATE 06/30/92

SVL ANALYTICAL

Client: ECHO BAY MINERALS, INC.
Client Sample ID: LF-1
Sample Collection Date: 07/08/92 12:30 PM
Date of Sample Receipt: 07/14/92
Date of Report: 7/28/92

SVL Job Number: 20797
SVL Sample ID: W27177
Area:
Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	ANALYTE	RESULT	UNIT	DILN
pH	8.08	units		Molybdenum-D	< .02	mg/L	
Alkalinity	246.	mg/L		Nickel-D	< .02	mg/L	
Bicarbonate	246.	mg/L		Potassium-D	3.	mg/L	
WAD Cyanide	< .01	mg/L		Selenium-D	< .002	mg/L	
Aluminum-D	.08	mg/L		Silver-D	< .006	mg/L	
Arsenic-D	< .002	mg/L		Sodium-D	9.71	mg/L	
Barium-D	.035	mg/L		Zinc-D	.016	mg/L	
Boron-D	< .02	mg/L		Chloride	3.2	mg/L	
Cadmium-D	< .003	mg/L		Fluoride	.3	mg/L	
Calcium-D	127.30	mg/L		Sulfate	189.0	mg/L	
Chromium-D	< .005	mg/L		NO3+NO2-N	.74	mg/L	
Copper-D	.056	mg/L		o-Phosphate	< .01	mg/L	
Iron-D	.12	mg/L		Conductance	793.	umhos	
Lead-D	< .001	mg/L		TDS	521.	mg/L	
Magnesium-D	32.56	mg/L		TSS	53.0	mg/L	
Manganese-D	.007	mg/L		Oil & Grease	--NA--		
Mercury-D	< .0002	mg/L					

--NA-- NOT ANALYZED

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

M. Bencich by su DATE 07/29/92

SVL ANALYTICAL

Client: ECHO BAY MINERALS, INC.
Client Sample ID: LF-2
Sample Collection Date: 07/08/92 12:15 PM
Date of Sample Receipt: 07/14/92
Date of Report: 7/28/92

SVL Job Number: 20797
SVL Sample ID: W27178
Area:
Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	ANALYTE	RESULT	UNIT	DILN
pH	8.00	units		Molybdenum-D	< .02	mg/L	
Alkalinity	287.	mg/L		Nickel-D	< .02	mg/L	
Bicarbonate	287.	mg/L		Potassium-D	3.	mg/L	
WAD Cyanide	< .01	mg/L		Selenium-D	< .002	mg/L	
Aluminum-D	.09	mg/L		Silver-D	< .006	mg/L	
Arsenic-D	< .002	mg/L		Sodium-D	9.98	mg/L	
Barium-D	.034	mg/L		Zinc-D	.019	mg/L	
Boron-D	< .02	mg/L		Chloride	4.6	mg/L	
Cadmium-D	< .003	mg/L		Fluoride	.3	mg/L	
Calcium-D	173.50	mg/L		Sulfate	286.0	mg/L	
Chromium-D	< .005	mg/L		NO3+NO2-N	.48	mg/L	
Copper-D	.007	mg/L		o-Phosphate	< .01	mg/L	
Iron-D	.16	mg/L		Conductance	994.	umhos	
Lead-D	< .001	mg/L		TDS	699.	mg/L	
Magnesium-D	46.50	mg/L		TSS	585.6	mg/L	
Manganese-D	.009	mg/L		Oil & Grease	--NA--		
Mercury-D	< .0002	mg/L					

--NA-- NOT ANALYZED

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

M. Bencich by DATE 07/29/92

Client: ECHO BAY MINERALS, INC.
Client Sample ID: LF-2
Sample Collection Date: 08/05/92 NOON
Date of Sample Receipt: 08/07/92
Date of Report: 8/24/92

SVL Job Number: 20937
SVL Sample ID: W28555
Area:
Matrix: WATER

ANALYTE		RESULT	UNIT	DILN
Silver-D	<	.004	mg/L	
Aluminum-D		.39	mg/L	
Arsenic-D	<	.035	mg/L	
Boron-D		.01	mg/L	
Barium-D		.033	mg/L	
Calcium-D		147.00	mg/L	
Cadmium-D	<	.003	mg/L	
Chromium-D	<	.004	mg/L	
Copper-D		.005	mg/L	
Iron-D		.65	mg/L	
Potassium-D		3.	mg/L	
Magnesium-D		41.30	mg/L	
Manganese-D		.012	mg/L	
Molybdenum-D	<	.008	mg/L	
Sodium-D		9.91	mg/L	
Nickel-D	<	.010	mg/L	
Lead-D	<	.041	mg/L	

ANALYTE		RESULT	UNIT	DILN
Selenium-D	<	.064	mg/L	
Zinc-D		.011	mg/L	
Mercury-D	<	.0002	mg/L	
Chloride		4.7	mg/L	
Fluoride		290.0	mg/L	
Sulfate		.3	mg/L	
Alkalinity		259.	mg/L	
Conductance		955.	umhos	
TDS		1414	mg/L	
TSS		3096	mg/L	
Oil & Grease		--NA--		
o-Phosphate		4.72	mg/L	
Bicarbonate		259.	mg/L	
pH		7.85	units	
NO3+NO2-N		.40	mg/L	
WAD Cyanide	<	.01	mg/L	

--NA-- NOT ANALYZED

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

Reeba Deacich my dc DATE 08/25/92

Client: ECHO BAY MINERALS, INC.
Client Sample ID: LF-2
Sample Collection Date: 09/08/92 10:00 AM
Date of Sample Receipt: 09/11/92
Date of Report: 9/29/92

SVL Job Number: 21163
SVL Sample ID: W30853
Area:
Matrix: WATER

ANALYTE		RESULT	UNIT	DILN	ANALYTE		RESULT	UNIT	DILN
pH		7.88	units		Molybdenum-D	<	.02	mg/L	
Alkalinity		265.	mg/L		Nickel-D	<	.02	mg/L	
Bicarbonate		265.	mg/L		Potassium-D		3.	mg/L	
WAD Cyanide	<	.01	mg/L		Selenium-D	<	.002	mg/L	
Aluminum-D	<	.04	mg/L		Silver-D	<	.006	mg/L	
Arsenic-D		.035	mg/L		Sodium-D		11.68	mg/L	
Barium-D		.032	mg/L		Zinc-D		.006	mg/L	
Boron-D	<	.02	mg/L		Chloride		5.7	mg/L	
Cadmium-D	<	.003	mg/L		Fluoride		.3	mg/L	
Calcium-D		166.60	mg/L		Sulfate		326.0	mg/L	
Chromium-D	<	.005	mg/L		NO3+NO2-N		.41	mg/L	
Copper-D	<	.005	mg/L		o-Phosphate		3.30	mg/L	
Iron-D		.04	mg/L		Conductance		1020	umhos	
Lead-D		.012	mg/L		TDS		746.	mg/L	
Magnesium-D		48.16	mg/L		TSS		930.0	mg/L	
Manganese-D		.002	mg/L		Oil & Grease		--NA--		
Mercury-D	<	.0002	mg/L						

--NA-- NOT ANALYZED

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

by AC DATE 09/29/92

SVL ANALYTICAL

Client: ECHO BAY MINERALS, INC.
Client Sample ID: LF-1
Sample Collection Date: 10/14/92 10:30AM
Date of Sample Receipt: 10/19/92
Date of Report: 11/16/92 REVISED

SVL Job Number: 21387
SVL Sample ID: W33388
Area:
Matrix: WATER

ANALYTE		RESULT	UNIT	DILN	ANALYTE		RESULT	UNIT	DILN
Silver	<	.004	mg/L		Selenium	<	.062	mg/L	
Aluminum		10.3	mg/L		Zinc		.130	mg/L	
Arsenic		.065	mg/L		Mercury	<	.0002	mg/L	
Boron		.042	mg/L		Chloride		2.54	mg/L	
Barium		.204	mg/L		Fluoride		.28	mg/L	
Calcium		145.1	mg/L		Sulfate		218.3	mg/L	
Cadmium	<	.004	mg/L		Alkalinity		274.	mg/L	
Chromium		.023	mg/L		Conductance		865.	mg/L	
Copper		.076	mg/L		TDS		610.	mg/L	
Iron		22.8	mg/L		TSS		434.7	mg/L	
Potassium		4.39	mg/L		Oil & Grease		--NA--		
Magnesium		41.5	mg/L		o-Phosphate		.10	mg/L	
Manganese		.458	mg/L		Bicarbonate		274.	mg/L	
Molybdenum		.008	mg/L		pH		8.0	units	
Sodium		11.5	mg/L		NO3+NO2-N		.66	mg/L	
Nickel		.035	mg/L		WAD Cyanide	<	.01	mg/L	
Lead	<	.049	mg/L						

--NA-- NOT ANALYZED

DISSOLVED METALS

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

DATE 11/16/92

SVL ANALYTICAL

Client: ECHO BAY MINERALS, INC.
Client Sample ID: LF-2
Sample Collection Date: 10/14/92 10:45AM
Date of Sample Receipt: 10/19/92
Date of Report: 11/16/92 REVISED

SVL Job Number: 21387
SVL Sample ID: W33389
Area:
Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	ANALYTE	RESULT	UNIT	DILN
Silver	< .004	mg/L		Selenium	< .062	mg/L	
Aluminum	22.3	mg/L		Zinc	.257	mg/L	
Arsenic	.108	mg/L		Mercury	.0002	mg/L	
Boron	.066	mg/L		Chloride	4.68	mg/L	
Barium	.347	mg/L		Fluoride	.32	mg/L	
Calcium	174.7	mg/L		Sulfate	265.2	mg/L	
Cadmium	.005	mg/L		Alkalinity	300.	mg/L	
Chromium	.049	mg/L		Conductance	976.	mg/L	
Copper	.213	mg/L		TDS	716.	mg/L	
Iron	48.4	mg/L		TSS	473.3	mg/L	
Potassium	6.85	mg/L		Oil & Grease	--NA--		
Magnesium	53.2	mg/L		o-Phosphate	.11	mg/L	
Manganese	.676	mg/L		Bicarbonate	300.	mg/L	
Molybdenum	< .008	mg/L		pH	8.0	units	
Sodium	16.6	mg/L		NO3+NO2-N	.50	mg/L	
Nickel	.055	mg/L		WAD Cyanide	< .01	mg/L	
Lead	< .049	mg/L					

--NA-- NOT ANALYZED

DISSOLVED METALS

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

Kelli Dencich by dc DATE 11/16/92

SVL ANALYTICAL

Client: ECHO BAY
Client Sample ID: LF-1
Sample Collection Date: 11/10/92 11:30AM
Date of Sample Receipt: 11/16/92
Date of Report: 12/02/92

SVL Job Number: 21503
SVL Sample ID: W34649
Area:
Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	M	ANALYTE	RESULT	UNIT	DILN	M
pH	7.80	units			Molybdenum-D	< .007	mg/L		P
Alkalinity	266.	mg/L			Nickel-D	< .02	mg/L		P
Bicarbonate	266.	mg/L			Potassium-D	3.16	mg/L		P
WAD Cyanide	< .01	mg/L		C	Selenium-D	< .040	mg/L		P
Aluminum-D	.60	mg/L		P	Silver-D	.003	mg/L		P
Arsenic-D	< .04	mg/L		P	Sodium-D	11.61	mg/L		P
Barium-D	.042	mg/L		P	Zinc-D	.014	mg/L		P
Boron-D	< .06	mg/L		P	Chloride	2.8	mg/L		I
Cadmium-D	< .002	mg/L		P	Fluoride	.3	mg/L		I
Calcium-D	140.40	mg/L		P	Sulfate	224.0	mg/L		I
Chromium-D	< .003	mg/L		P	NO3+NO2-N	.66	mg/L		C
Copper-D	.005	mg/L		P	o-Phosphate	4.63	mg/L		
Iron-D	.584	mg/L		P	Conductance	826.	umhos		
Lead-D	.10	mg/L		P	TDS	629.	mg/L		
Magnesium-D	37.22	mg/L		P	TSS	1737.3	mg/L		
Manganese-D	.027	mg/L		P	Oil & Grease,	--NA--			
Mercury-D	< .0002	mg/L		V					

M(ethod): P=ICP F=GFAA V=CVAA C=COLORIMETRIC I=ION CHROMATOGRAPHY
--NA-- : NOT ANALYZED

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

M. Bencich by SH DATE 12/02/92

SVL ANALYTICAL

Client: ECHO BAY
Client Sample ID: LF-2
Sample Collection Date: 11/10/92 11:15AM
Date of Sample Receipt: 11/16/92
Date of Report: 12/02/92

SVL Job Number: 21503
SVL Sample ID: W34650
Area:
Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	M	ANALYTE	RESULT	UNIT	DILN	M
pH	7.80	units			Molybdenum-D	< .007	mg/L		P
Alkalinity	285.	mg/L			Nickel-D	< .02	mg/L		P
Bicarbonate	285.	mg/L			Potassium-D	2.47	mg/L		P
WAD Cyanide	< .01	mg/L		C	Selenium-D	< .040	mg/L		P
Aluminum-D	.14	mg/L		P	Silver-D	< .002	mg/L		P
Arsenic-D	< .04	mg/L		P	Sodium-D	15.67	mg/L		P
Barium-D	.036	mg/L		P	Zinc-D	.016	mg/L		P
Boron-D	< .06	mg/L		P	Chloride	4.3	mg/L		I
Cadmium-D	< .002	mg/L		P	Fluoride	.3	mg/L		I
Calcium-D	158.60	mg/L		P	Sulfate	275.0	mg/L		I
Chromium-D	< .003	mg/L		P	NO3+NO2-N	.44	mg/L		C
Copper-D	.005	mg/L		P	o-Phosphate	1.19	mg/L		
Iron-D	.183	mg/L		P	Conductance	956.	umhos		
Lead-D	< .04	mg/L		P	TDS	715.	mg/L		
Magnesium-D	41.36	mg/L		P	TSS	638.3	mg/L		
Manganese-D	.007	mg/L		P	Oil & Grease.	--NA--			
Mercury-D	< .0002	mg/L		V					

M(ethod): P=ICP F=GFAA V=CVAA C=COLORIMETRIC I=ION CHROMATOGRAPHY
--NA-- : NOT ANALYZED

THIS REPORT HAS BEEN CHECKED AND IS CERTIFIED TO BE ACCURATE:

M. Bencich by SH DATE 12/02/92

/93 — Water Analysis Report 12:23 |

Page 1

ient : Echo Bay Minerals, Inc.
: ss : 921 Fish Hatchery Road
Republic, Washington 99166
ttn. : Accounts Payable cc: George Swank
roject :
ample Matrix:
ample ID: LF1
ample Date Time: 10/06/93 12:45

Lab No. : 93-WI/10440
Date Received: 10/08/93

Parameters		
Alkalinity as CaCO3	282.	mg/l
Bicarbonate as CaCO3	282	mg/l
Carbonate as CaCO3	0.	mg/l
Calcium, dissolved	122.	mg/l
Chloride	3.	mg/l
Conductivity @ 25C	755.	umhos/cm
Cyanide, total	0.006	mg/l
pH (lab)	7.3	units
Hardness as CaCO3	432.	mg/l
Magnesium, dissolved	31.	mg/l
Nitrate as N, dissolved	0.77	mg/l
Nitrate/Nitrite as N	0.77	mg/l
Nitrite as N, dissolved	-0.01	mg/l
Potassium, dissolved	2.	mg/l
Phosphorus, ortho diss	-0.005	mg/l
Sodium, dissolved	10.	mg/l
Sulfate	177.	mg/l
Turbidity	6.	ntu
TDS Ratio (grav./calc.)	1.03	
Solids, total dissolved	526.	mg/l
Solids, total suspended	26.	mg/l
Cation-Anion Balance	-1.67	%
Arsenic, dissolved	0.007	mg/l
Barium, dissolved	0.03	mg/l
Cadmium, dissolved	-0.005	mg/l
Chromium, dissolved	-0.01	mg/l
Copper, dissolved	-0.01	mg/l
Lead, dissolved	-0.02	mg/l
Manganese, dissolved	-0.01	mg/l
Mercury, dissolved	-0.0002	mg/l
Molybdenum, dissolved	-0.01	mg/l
Nickel, dissolved	-0.02	mg/l
Selenium, dissolved	-0.001	mg/l
Silver, dissolved	-0.01	mg/l
Zinc, dissolved	0.02	mg/l

marks:
te: Negative sign "-" denotes that the value is less than "<"

ott Habermahl, Project Manager/SH

ank E. Polniak, Inorganic Laboratory Supervisor/

ient : Echo Bay Minerals, Inc.
ss : 921 Fish Hatchery Road
Republic, Washington 99166
: Accounts Payable cc: George Swank
Project :
Sample Matrix:
Sample ID: LF2
Sample Date Time: 10/06/93 12:30

Lab No. : 93-WI/10441
Date Received: 10/08/93

Parameters

Alkalinity as CaCO3	302.	mg/l
Bicarbonate as CaCO3	302	mg/l
Carbonate as CaCO3	0.	mg/l
Calcium, dissolved	150.	mg/l
Chloride	3.	mg/l
Conductivity @ 25C	877.	umhos/cm
Cyanide, total	0.006	mg/l
pH (lab)	7.6	units
Hardness as CaCO3	539.	mg/l
Magnesium, dissolved	40.	mg/l
Nitrate as N, dissolved	1.01	mg/l
Nitrate/Nitrite as N	1.01	mg/l
Nitrite as N, dissolved	-0.01	mg/l
Potassium, dissolved	3.	mg/l
Phosphorus, ortho diss	-0.005	mg/l
Sodium, dissolved	11.	mg/l
Sulfate	263.	mg/l
Turbidity	0.47	ntu
TDS Ratio (grav./calc.)	1.03	
Solids, total dissolved	666.	mg/l
Solids, total suspended	-2.	mg/l
Cation-Anion Balance	-1.35	%
Arsenic, dissolved	0.005	mg/l
Barium, dissolved	0.03	mg/l
Cadmium, dissolved	-0.005	mg/l
Chromium, dissolved	-0.01	mg/l
Copper, dissolved	-0.01	mg/l
Lead, dissolved	-0.02	mg/l
Manganese, dissolved	-0.01	mg/l
Mercury, dissolved	-0.0002	mg/l
Molybdenum, dissolved	-0.01	mg/l
Nickel, dissolved	-0.02	mg/l
Selenium, dissolved	-0.001	mg/l
Silver, dissolved	-0.01	mg/l
Zinc, dissolved	-0.01	mg/l

emarks:
ote: Negative sign "-" denotes that the value is less than "<"

cott Habermehl, Project Manager/SH.

rank E. Polniak, Inorganic Laboratory Supervisor/

BORATORIES INC DATA MANAGEMENT SYSTEM !

2/93 Water Analysis Report 11:37 !

Page 1

nt : Echo Bay Minerals Company
dress : 921 Fish Hatchery Rd.
Republic, WA 99166
Attn. : Mr. W. Hunt
Project :
Sample ID: LF-1 Lab No. : 93-WI/11370
Sample Date Time: 11/02/93 13:15 Date Received: 11/05/93

Parameters		
Alkalinity as CaCO3	270.	mg/l
Bicarbonate as CaCO3	270	mg/l
Calcium, dissolved	119.	mg/l
Carbonate as CaCO3	0.	mg/l
Chloride	2.	mg/l
Conductivity @ 25C	782.	umhos/cm
Magnesium, dissolved	32.	mg/l
Nitrate as N, dissolved	0.70	mg/l
Nitrate/Nitrite as N	0.70	mg/l
Nitrite as N, dissolved	-0.01	mg/l
pH (lab)	8.1	units
Potassium, dissolved	3.	mg/l
Phosphorus, ortho diss	-0.005	mg/l
Sodium, dissolved	13.	mg/l
Sulfate	200.	mg/l
Cation-Anion Balance	-2.33	%
Solids, total dissolved	554.	mg/l
Solids, total suspended	8.	mg/l
Turbidity	1.94	ntu
Arsenic, dissolved	0.008	mg/l
Barium, dissolved	0.03	mg/l
Cadmium, dissolved	-0.005	mg/l
Chromium, dissolved	-0.01	mg/l
Copper, dissolved	-0.01	mg/l
Lead, dissolved	-0.02	mg/l
Manganese, dissolved	-0.01	mg/l
Mercury, dissolved	-0.0002	mg/l
Molybdenum, dissolved	-0.01	mg/l
Nickel, dissolved	-0.02	mg/l
Selenium, dissolved	-0.001	mg/l
Silver, dissolved	-0.01	mg/l
Zinc, dissolved	-0.01	mg/l

Remarks:
Note: Negative sign "-" denotes that the value is less than "<"

Scott Habermehl, Project Manager/SH.

Frank E. Polniak, Inorganic Laboratory Supervisor/FP

RATORIES INC　　　　　　　　DATA MANAGEMENT SYSTEM

93　　　Water Analysis Report　　　11:38

Page 1

nt : Echo Bay Minerals Company
ress : 921 Fish Hatchery Rd.
Republic, WA 99166
ttn. : Mr. W. Hunt
Project :

Sample ID: LF-2　　　Lab No. : 93-WI/11371
Sample Date Time: 11/02/93 13:00　　　Date Received: 11/05/93

Parameters		
Alkalinity as CaCO3	296.	mg/l
Bicarbonate as CaCO3	296	mg/l
Calcium, dissolved	150.	mg/l
Carbonate as CaCO3	0.	mg/l
Chloride	3.	mg/l
Conductivity @ 25C	968.	umhos/cm
Magnesium, dissolved	41.	mg/l
Nitrate as N, dissolved	1.01	mg/l
Nitrate/Nitrite as N	1.01	mg/l
Nitrite as N, dissolved	-0.01	mg/l
pH (lab)	8.0	units
Potassium, dissolved	3.	mg/l
Phosphorus, ortho diss	-0.005	mg/l
Sodium, dissolved	13.	mg/l
Sulfate	278.	mg/l
Cation-Anion Balance	-1.41	%
Solids, total dissolved	710.	mg/l
Solids, total suspended	-2.	mg/l
Turbidity	0.20	ntu
Arsenic, dissolved	0.005	mg/l
Barium, dissolved	0.03	mg/l
Cadmium, dissolved	-0.005	mg/l
Chromium, dissolved	-0.01	mg/l
Copper, dissolved	-0.01	mg/l
Lead, dissolved	-0.02	mg/l
Manganese, dissolved	-0.01	mg/l
Mercury, dissolved	-0.0002	mg/l
Molybdenum, dissolved	-0.01	mg/l
Nickel, dissolved	-0.02	mg/l
Selenium, dissolved	-0.001	mg/l
Silver, dissolved	-0.01	mg/l
Zinc, dissolved	-0.01	mg/l

Remarks:
Note: Negative sign "-" denotes that the value is less than "<"

Scott Habermehl, Project Manager/SH.

Frank E. Polniak, Inorganic Laboratory Supervisor

ATORIES INC

DATA MANAGEMENT SYSTEM |

Water Analysis Report

13:19 |

Page 1

: Echo Bay Minerals, Inc.
ss : 921 Fish Hatchery Road
Republic, Washington 99166
n. : Accounts Payable cc: George Swank
oject :
ample Matrix:
Sample ID: LF-1
Sample Date Time: 12/07/93 12:30

Lab No. : 93-WI/12399
Date Received: 12/10/93

Parameters

Alkalinity as CaCO3	262.	mg/l
Bicarbonate as CaCO3	262	mg/l
Carbonate as CaCO3	0.	mg/l
Calcium, dissolved	118.	mg/l
Chloride	3.	mg/l
Conductivity @ 25C	784.	umhos/cm
pH (lab)	8.2	units
Hardness as CaCO3	426.	mg/l
Magnesium, dissolved	32.	mg/l
Nitrate as N, dissolved	0.68	mg/l
Nitrate/Nitrite as N	0.68	mg/l
Nitrite as N, dissolved	-0.01	mg/l
Nitrate as NO3	3.0	mg/l
Potassium, dissolved	2.	mg/l
Phosphorus, ortho diss	0.005	mg/l
Sodium, dissolved	12.	mg/l
Sulfate	179.	mg/l
SAR in water	.26	
Turbidity	0.9	ntu
TDS Ratio (grav./calc.)	.98	
Solids, total dissolved	490.	mg/l
Solids, total suspended	-2.	mg/l
Cation-Anion Balance	.11	%
Arsenic, dissolved	0.007	mg/l
Barium, dissolved	0.03	mg/l
Cadmium, dissolved	-0.005	mg/l
Chromium, dissolved	-0.01	mg/l
Copper, dissolved	-0.01	mg/l
Lead, dissolved	-0.02	mg/l
Manganese, dissolved	-0.01	mg/l
Mercury, dissolved	-0.0002	mg/l
Molybdenum, dissolved	-0.01	mg/l
Nickel, dissolved	-0.02	mg/l
Selenium, dissolved	-0.001	mg/l
Silver, dissolved	-0.01	mg/l
Zinc, dissolved	-0.01	mg/l

emarks:
ote: Negative sign "-" denotes that the value is less than "<"

cott Habermehl, Project Manager /# for SH

r[illegible]k E. Polniak, Inorganic Laboratory Supervisor /

OPIES INC

DATA MANAGEMENT SYSTEM |
13:20 |

Water Analysis Report

Page 1

: Echo Bay Minerals, Inc.
: 921 Fish Hatchery Road
Republic, Washington 99166
: Accounts Payable cc: George Swank
ject :
mple Matrix:
ample ID: LF-2
Sample Date Time: 12/07/93 12:15

Lab No. : 93-WI/12396
Date Received: 12/10/93

Parameters

Alkalinity as CaCO3	282.	mg/l
Bicarbonate as CaCO3	282	mg/l
Carbonate as CaCO3	0.	mg/l
Calcium, dissolved	132.	mg/l
Chloride	3.	mg/l
Conductivity @ 25C	873.	umhos/cm
pH (lab)	8.2	units
Hardness as CaCO3	486.	mg/l
Magnesium, dissolved	38.	mg/l
Nitrate as N, dissolved	1.26	mg/l
Nitrate/Nitrite as N	1.26	mg/l
Nitrite as N, dissolved	-0.01	mg/l
Nitrate as NO3	5.5	mg/l
Potassium, dissolved	2.	mg/l
Phosphorus, ortho diss	0.006	mg/l
Sodium, dissolved	12.	mg/l
Sulfate	218.	mg/l
SAR in water	.24	
Turbidity	6.1	ntu
TDS Ratio (grav./calc.)	1.05	
Solids, total dissolved	602.	mg/l
Solids, total suspended	16.	mg/l
Cation-Anion Balance	0.00	%
Arsenic, dissolved	0.005	mg/l
Barium, dissolved	0.03	mg/l
Cadmium, dissolved	-0.005	mg/l
Chromium, dissolved	-0.01	mg/l
Copper, dissolved	-0.01	mg/l
Lead, dissolved	-0.02	mg/l
Manganese, dissolved	-0.01	mg/l
Mercury, dissolved	-0.0002	mg/l
Molybdenum, dissolved	-0.01	mg/l
Nickel, dissolved	-0.02	mg/l
Selenium, dissolved	-0.001	mg/l
Silver, dissolved	-0.01	mg/l
Zinc, dissolved	-0.01	mg/l

emarks:
ote: Negative sign "-" denotes that the value is less than "<"

cott Habermehl, Project Manager/st for SH

rank E. Polniak, Inorganic Laboratory Supervisor/

ACZ LABORATORIES INC DATA MANAGEMENT SYSTEM I
02/24/94 Water Analysis Report 15:32 I
== Page 1 ========

lient : Echo Bay Minerals, Inc.
ldress : 921 Fish Hatchery Road
Republic, Washington 99166
Attn. : Accounts Payable cc: George Swank
Project :
Sample ID: LF-1 Lab No. : 94-WI/00842
Sample Date Time: 02/08/94 12:00 Date Received: 02/10/94

Parameters		
Alkalinity as CaCO3	284.	mg/l
Bicarbonate as CaCO3	284	mg/l
Calcium, dissolved	119.	mg/l
Carbonate as CaCO3	0.	mg/l
Chloride	2.	mg/l
Conductivity @ 25C	800.	umhos/cm
Magnesium, dissolved	32.	mg/l
Nitrate as N, dissolved	0.69	mg/l
Nitrate/Nitrite as N	0.69	mg/l
Nitrite as N, dissolved	-0.01	mg/l
pH (lab)	7.7	units
Potassium, dissolved	-5.	mg/l
Phosphorus, ortho diss	-0.005	mg/l
Sodium, dissolved	12.	mg/l
Sulfate	173.	mg/l
Cation-Anion Balance	-2.18	%
Solids, total dissolved	542.	mg/l
Solids, total suspended	-2.	mg/l
Turbidity	0.7	ntu
Arsenic, dissolved	0.006	mg/l
Barium, dissolved	-0.05	mg/l
Cadmium, dissolved	-0.025	mg/l
Chromium, dissolved	-0.05	mg/l
Copper, dissolved	-0.05	mg/l
Lead, dissolved	-0.10	mg/l
Manganese, dissolved	-0.05	mg/l
Mercury, dissolved	-0.0002	mg/l
Molybdenum, dissolved	-0.05	mg/l
Nickel, dissolved	-0.10	mg/l
Selenium, dissolved	-0.001	mg/l
Silver, dissolved	-0.05	mg/l
Zinc, dissolved	-0.05	mg/l

Remarks:

Note: Negative sign "-" denotes that the value is less than "<"

Scott Habermehl, Project Manager/S.H.

Frank E. Polniak, Inorganic Laboratory Supervisor

LABORATORIES INC DATA MANAGEMENT SYSTEM

02/25/94 Water Analysis Report 16:58

Page 1

Client : Echo Bay Minerals, Inc.
Address : 921 Fish Hatchery Road
Republic, Washington 99166
Attn. : Accounts Payable cc: George Swank
Project :
Sample ID: LF-2
Sample Date Time: 02/08/94 11:45

Lab No. : 94-WI/00843
Date Received: 02/10/94

Parameters		
Alkalinity as CaCO3	312.	mg/l
Bicarbonate as CaCO3	312	mg/l
Calcium, dissolved	141.	mg/l
Carbonate as CaCO3	0.	mg/l
Chloride	3.	mg/l
Conductivity @ 25C	900.	umhos/cm
Magnesium, dissolved	35.	mg/l
Nitrate as N, dissolved	0.73	mg/l
Nitrate/Nitrite as N	0.73	mg/l
Nitrite as N, dissolved	-0.01	mg/l
pH (lab)	7.6	units
Potassium, dissolved	3.	mg/l
Phosphorus, ortho diss	-0.005	mg/l
Sodium, dissolved	13.	mg/l
Sulfate	234.	mg/l
Cation-Anion Balance	-3.07	%
Solids, total dissolved	642.	mg/l
Solids, total suspended	78.	mg/l
Turbidity	19.0	ntu
Arsenic, dissolved	0.007	mg/l
Barium, dissolved	0.03	mg/l
Cadmium, dissolved	0.006	mg/l
Chromium, dissolved	-0.01	mg/l
Copper, dissolved	-0.01	mg/l
Lead, dissolved	-0.02	mg/l
Manganese, dissolved	0.02	mg/l
Mercury, dissolved	-0.0002	mg/l
Molybdenum, dissolved	-0.01	mg/l
Nickel, dissolved	-0.02	mg/l
Selenium, dissolved	-0.001	mg/l
Silver, dissolved	-0.01	mg/l
Zinc, dissolved	0.03	mg/l

Remarks:
Note: Negative sign "-" denotes that the value is less than "<"

Scott Habermehl, Project Manager

Frank E. Polniak, Inorganic Laboratory Supervisor/S.H.

ATORIES INC DATA MANAGEMENT SYSTEM
Water Analysis Report Page 2 of 2 17:24
=== Page 1 =========

: Echo Bay Minerals, Inc.
: 921 Fish Hatchery Road
Republic, Washington 99166
: Accounts Payable cc: George Swank
ject :

mple ID: LF-1 Lab No. : 94-WI/01742
mple Date Time: 03/08/94 12:30 Date Received: 03/12/94

Parameters

Lead, dissolved	-0.02	mg/l
Manganese, dissolved	-0.01	mg/l
Mercury, dissolved	-0.0002	mg/l
Molybdenum, dissolved	-0.01	mg/l
Nickel, dissolved	-0.02	mg/l
Selenium, dissolved	0.001	mg/l
Silver, dissolved	-0.01	mg/l
Zinc, dissolved	-0.02	mg/l

Remarks:
Note: Negative sign "-" denotes that the value is less than "<"

Scott Habermehl, Project Manager/SH.

Frank E. Polniak, Inorganic Laboratory Supervisor/W

TORIES INC 17:22 1

Water Analysis Report Page 1 of 2

Echo Bay Minerals, Inc.
921 Fish Hatchery Road
Republic, Washington 99166
Accounts Payable cc: George Swank

ID: LF-2
Date Time: 03/08/94 12:15

Lab No. : 94-WI/01743
Date Received: 03/12/94

Parameters

Alkalinity as CaCO3	306.	mg/l
Bicarbonate as CaCO3	306	mg/l
Calcium, dissolved	153.	mg/l
Carbonate as CaCO3	0.	mg/l
Chloride	3.	mg/l
Hardness as CaCO3	538.	mg/l
Magnesium, dissolved	38.	mg/l
Nitrate as N, dissolved	0.87	mg/l
Nitrate/Nitrite as N	0.87	mg/l
Nitrite as N, dissolved	-0.01	mg/l
pH (lab)	8.0	units
Conductivity @ 25C	967.	umhos/cm
Potassium, dissolved	3.	mg/l
Sodium, dissolved	13.	mg/l
Sulfate	249.	mg/l
Cations (sum)	11.42	meq/l
Anions (sum)	11.43	meq/l
Cation-Anion Balance	-.04	%
Solids, total dissolved	648.	mg/l
Solids, total suspended	2.	mg/l
Arsenic, dissolved	0.006	mg/l
Barium, dissolved	0.03	mg/l
Cadmium, dissolved	-0.005	mg/l
Chromium, dissolved	-0.01	mg/l
Copper, dissolved	-0.01	mg/l

Note: Negative sign "-" denotes that the value is less than "<"

Scott Habermehl, Project Manager/SH.

Frank E. Polniak, Inorganic Laboratory Supervisor/

Water Analysis Report

cho Bay Minerals, Inc.
21 Fish Hatchery Road
Republic, Washington 99166
Accounts Payable cc: George Swank

e :
le ID: LF 1
mple Date Time: 04/06/94 11:45

Lab No. : 94-W1/02697
Date Received: 04/08/94

Parameters		
Alkalinity as CaCO3	258.	mg/l
Bicarbonate as CaCO3	258	mg/l
Calcium, dissolved	120.	mg/l
Carbonate as CaCO3	0.	mg/l
Chloride	1.	mg/l
Conductivity @ 25C	766.	umhos/cm
Magnesium, dissolved	32.	mg/l
Nitrate as N, dissolved	1.07	mg/l
Nitrate/Nitrite as N	1.07	mg/l
Nitrite as N, dissolved	-0.01	mg/l
pH (lab)	7.8	units
Potassium, dissolved	2.	mg/l
Phosphorus, ortho diss	-0.005	mg/l
Sodium, dissolved	12.	mg/l
Sulfate	152.	mg/l
Cation-Anion Balance	4.66	%
Solids, total dissolved	478.	mg/l
Solids, total suspended	10.	mg/l
Turbidity	3.8	ntu
Arsenic, dissolved	0.006	mg/l
Barium, dissolved	0.03	mg/l
Cadmium, dissolved	-0.005	mg/l
Chromium, dissolved	-0.01	mg/l
Copper, dissolved	-0.01	mg/l
Lead, dissolved	-0.02	mg/l
Manganese, dissolved	-0.01	mg/l
Mercury, dissolved	-0.0002	mg/l
Molybdenum, dissolved	-0.01	mg/l
Nickel, dissolved	-0.02	mg/l
Selenium, dissolved	-0.001	mg/l
Silver, dissolved	-0.01	mg/l
Zinc, dissolved	-0.01	mg/l

Remarks:
Note: Negative sign "-" denotes that the value is less than "<"

Scott Habermehl, Project Manager/S.H.

Frank E. Polniak, Inorganic Laboratory Supervisor/

: Echo Bay Minerals, Inc.
: 921 Fish Hatchery Road
Republic, Washington 99166
n. : Accounts Payable cc: George Swank
oject :
ample ID: LF 2
Sample Date Time: 04/06/94 11:30

Lab No. : 94-WI/02698
Date Received: 04/08/94

Parameters		
Alkalinity as CaCO3	284.	mg/l
Bicarbonate as CaCO3	284	mg/l
Calcium, dissolved	154.	mg/l
Carbonate as CaCO3	0.	mg/l
Chloride	3.	mg/l
Conductivity @ 25C	967.	umhos/cm
Magnesium, dissolved	44.	mg/l
Nitrate as N, dissolved	1.97	mg/l
Nitrate/Nitrite as N	1.97	mg/l
Nitrite as N, dissolved	-0.01	mg/l
pH (lab)	7.7	units
Potassium, dissolved	3.	mg/l
Phosphorus, ortho diss	-0.005	mg/l
Sodium, dissolved	13.	mg/l
Sulfate	247.	mg/l
Cation-Anion Balance	4.41	%
Solids, total dissolved	670.	mg/l
Solids, total suspended	-2.	mg/l
Turbidity	-0.1	ntu
Arsenic, dissolved	0.004	mg/l
Barium, dissolved	0.03	mg/l
Cadmium, dissolved	-0.005	mg/l
Chromium, dissolved	-0.01	mg/l
Copper, dissolved	-0.01	mg/l
Lead, dissolved	-0.02	mg/l
Manganese, dissolved	-0.01	mg/l
Mercury, dissolved	-0.0002	mg/l
Molybdenum, dissolved	-0.01	mg/l
Nickel, dissolved	-0.02	mg/l
Selenium, dissolved	-0.001	mg/l
Silver, dissolved	-0.01	mg/l
Zinc, dissolved	-0.01	mg/l

Remarks:
Note: Negative sign "-" denotes that the value is less than "<"

Scott Habermehl, Project Manager/SH.

Frank E. Polniak, Inorganic Laboratory Supervisor/

APPENDIX G-2
INFILTRATION POND LF-I

t : Echo Bay Minerals, Inc.
ress : 921 Fish Hatchery Road
Republic, Washington 99166
ttn. : Accounts Payable cc: George Swank
r :t :

ample ID: LF-I
ample Date Time: 10/06/93 13:00

Lab No. : 93-WI/10448
Date Received: 10/08/93

Parameters		
Nitrate as N, dissolved	59.80	mg/l
Nitrate/Nitrite as N	65.2	mg/l
Nitrite as N, dissolved	5.4	mg/l
Copper, total recov.	0.01	mg/l
Lead, total recov.	-0.02	mg/l
Mercury, total recov.	-0.0002	mg/l
Zinc, total recov.	0.06	mg/l

emarks:
ote: Negative sign "-" denotes that the value is less than "<"

cott Habermehl, Project Manager/SH.

rank E. Polniak, Inorganic Laboratory Supervisor/

LABORATORIES INC
/23/93 Water Analysis Report 11:35

Page 1

Client : Echo Bay Minerals Company
Address : 921 Fish Hatchery Rd.
Republic, WA 99166
Attn. : Mr. W. Hunt
Project :
Sample ID: LF-1
Sample Date Time: 11/02/93 13:30

Lab No. : 93-WI/11372
Date Received: 11/05/93

Parameters		
Alkalinity as CaCO3	208.	mg/l
Bicarbonate as CaCO3	208	mg/l
Calcium, dissolved	154.	mg/l
Carbonate as CaCO3	0.	mg/l
Chloride	72.	mg/l
Conductivity @ 25C	2150.	umhos/cm
Magnesium, dissolved	86.	mg/l
Nitrate as N, dissolved	136.00	mg/l
Nitrate/Nitrite as N	147.	mg/l
Nitrite as N, dissolved	11.	mg/l
pH (lab)	8.3	units
Potassium, dissolved	21.	mg/l
Phosphorus, ortho diss	0.049	mg/l
Sodium, dissolved	63.	mg/l
Sulfate	545.	mg/l
Cation-Anion Balance	1.26	%
Solids, total dissolved	978.	mg/l
Solids, total suspended	-2.	mg/l
Turbidity	0.63	ntu
Arsenic, dissolved	0.004	mg/l
Barium, dissolved	.04	mg/l
Cadmium, dissolved	-0.005	mg/l
Chromium, dissolved	-0.01	mg/l
Copper, dissolved	-0.01	mg/l
Lead, dissolved	-0.02	mg/l
Manganese, dissolved	0.05	mg/l
Mercury, dissolved	-0.0002	mg/l
Molybdenum, dissolved	0.03	mg/l
Nickel, dissolved	-0.02	mg/l
Selenium, dissolved	0.005	mg/l
Silver, dissolved	-0.01	mg/l
Zinc, dissolved	0.03	mg/l

Remarks:
Note: Negative sign "-" denotes that the value is less than "<"

Scott Habermehl, Project Manager/SH.

Frank E. Polniak, Inorganic Laboratory Supervisor/FP

IES INC [illegible]

Water Analysis Report Page 1 of 2 17:21

Page 1

cho Bay Minerals, Inc.
921 Fish Hatchery Road
Republic, Washington 99166
Accounts Payable cc: George Swank

e ID: LF-1
le Date Time: 03/08/94 12:30

Lab No. : 94-W1/01742
Date Received: 03/12/94

Parameters

Alkalinity as $CaCO_3$	274.	mg/l
Bicarbonate as $CaCO_3$	274	mg/l
Calcium, dissolved	120.	mg/l
Carbonate as $CaCO_3$	0.	mg/l
Chloride	2.	mg/l
Hardness as $CaCO_3$	431.	mg/l
Magnesium, dissolved	32.	mg/l
Nitrate as N, dissolved	0.88	mg/l
Nitrate/Nitrite as N	0.88	mg/l
Nitrite as N, dissolved	-0.01	mg/l
pH (lab)	8.1	units
Conductivity @ 25C	791.	umhos/cm
Potassium, dissolved	2.	mg/l
Sodium, dissolved	12.	mg/l
Sulfate	163.	mg/l
Cations (sum)	9.20	meq/l
Anions (sum)	8.96	meq/l
Cation-Anion Balance	1.32	%
Solids, total dissolved	520.	mg/l
Solids, total suspended	2.	mg/l
Arsenic, dissolved	0.007	mg/l
Barium, dissolved	0.03	mg/l
Cadmium, dissolved	-0.005	mg/l
Chromium, dissolved	-0.01	mg/l
Copper, dissolved	-0.01	mg/l

Note: Negative sign "-" denotes that the value is less than "<"

Scott Habermehl, Project Manager SH.

Frank E. Polniak, Inorganic Laboratory Supervisor

nt : Echo Bay Minerals, Inc.
ess : 921 Fish Hatchery Road
Republic, Washington 99166
tn. : Accounts Payable cc: George Swank
roject :

Sample ID: LF-I
Sample Date Time: 04/06/94 11:15

Lab No. : 94-WI/02704
Date Received: 04/08/94

Parameters		
Nitrate as N, dissolved	24.04	mg/l
Nitrate/Nitrite as N	26.4	mg/l
Nitrite as N, dissolved	2.36	mg/l
Copper, total recov.	0.05	mg/l
Lead, total recov.	-0.02	mg/l
Mercury, total recov.	-0.0002	mg/l
Zinc, total recov.	0.05	mg/l

Remarks:
Note: Negative sign "-" denotes that the value is less than "<"

Scott Habermehl, Project Manager/S.H.

Frank E. Polniak, Inorganic Laboratory Supervisor/

SVL ANALYTICAL
Formerly Silver Valley Laboratories, Inc.

SVL Job Number: 30578
SVL Sample ID: W40991

Client: ECHO BAY
Collection Date: 06/08/93 1:00 PM
Date of Receipt: 06/11/93
Date of Report: 06/30/93

Sample: LF-I

Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	M	ANALYTE	RESULT	UNIT	DILN	M
pH	7.17	units		W	Lead	<0.04	mg/L		P
Conductance	0.2	umhos			Zinc	0.035	mg/L		P
Copper	0.014	mg/L		P	Mercury	<0.0002	mg/L		V
NO3+NO2-N	61.7	mg/L		C	TSS	182	mg/L		W

M(ethods): P=200.7 F=GFAA V=245.2 C=COLORIMETRIC I=300.0 L=EPA600/2-78-054 T=TITRATION W=WET CHEM E=ELECTRODE

This report has been reviewed and is certified to be accurate:

Joe Costello

DATE 07/01/93

Joe Costello, Project Manager

SVL ANALYTICAL
Formerly Silver Valley Laboratories, Inc.

SVL Job Number: 30449
SVL Sample ID: W39870

Client: ECHO BAY
Collection Date: 05/11/93 11:00 AM
Date of Receipt: 05/14/93
Date of Report: 06/08/93

Sample: LF-I

Matrix: WATER

ANALYTE	RESULT	UNIT	DILN	M	ANALYTE	RESULT	UNIT	DILN	M
pH	7.56	units		W	Zinc	0.165	mg/L		P
Copper	0.035	mg/L		P	Conductance	1330	umhos		
Lead	<0.04	mg/L		P	TSS	358	mg/L		W
Mercury	<0.0002	mg/L		V	NO3-N	33.3	mg/L		

M(ethods): P=200.7 F=GFAA V=245.2 C=COLORIMETRIC I=300.0 L=EPA600/2-78-054 T=TITRATION W=WET CHEM E=ELECTRODE

This report has been reviewed and is certified to be accurate:

Joe Costello / DATE 06/03/93
Joe Costello, Project Manager

APPENDIX G-3

DATABASE TABLES AND GRAPHS FOR LF-I, LF-1, AND LF-2

Woodward-Clyde
Engineering & sciences applied to the earth & its environment

July 13, 1994

Mr. George Brown
U.S. Bureau of Land Management
1103 N. Fancher
Spokane WA 99212-1275

Subject: Excel database tables and graphs for LF-I, LF-1, and LF-2 Lamefoot Mine Project (WCC Project 23295 Task FEIS)

Dear George:

Enclosed are disk and hard copies of the spreadsheets, graphs, and explanatory notes prepared from the water quality monitoring data from stations LF-I (infiltration pond) and wells LF-1 and LF-2 at the Lamefoot site. Important features and advantages of the tables include the following:

1. The three tables (for LF-I, LF-1, and LF-2 data) are constructed in Excel 5.0, a readily available and powerful spreadsheet program. For ease in updating the spreadsheets and plotting future graphs, the graphs have been directly linked to and produced from the data spreadsheets.

2. Spreadsheets can be readily updated by inserting lines of data and updating the calculation formulas. Graphs can be updated and replotted by including the date and concentration ranges of the new data set.

3. Spreadsheets have been set up to automatically calculate maxima, minima, averages, and whether a parameter equals or exceeds state groundwater criteria.

4. Complete notes have been included on abbreviations used in the tables and assumptions used for calculations and graphs.

Key (indicator) parameters which are included in the tables include field pH, lab pH, field specific conductivity, lab specific conductivity, total alkalinity, arsenic, copper, iron, lead, manganese, zinc, sulfate, nitrate, total dissolved solids, and total suspended solids. Three parameters originally proposed are not included (aluminum, barium, and cadmium), because they were not on the short list of parameters to be monitored most frequently in the future. Where data sets of more than two points were available, graphs were constructed to display field pH; field specific conductivity; total alkalinity; total recoverable or dissolved iron, manganese, and zinc; sulfate; nitrate; and total dissolved solids.

We hope these spreadsheets will be useful in your discussions Friday and in monitoring future activities at the site. Please do not hesitate to call us if you have further questions. If you wish, we can arrange for your staffers who will be updating the spreadsheets and graphs to talk with the WCC staffers who constructed them.

Yours sincerely,

Sandra Lewellen Jones

Scott Mernitz
Project Manager

Sandra Lewellen Jones
Assistant Project Scientist

SJ:slj

(1 copy sent)

Enclosures

1c: Mr. Walt Hunt, Echo Bay Minerals Co.
Chris Paulsen
Bob Moran
Mark Zuber
Andre Fiedler
Project File 23295 FEIS

23295/LE19 07-13-94(3:59pm)/RPT/5

SUMMARY OF SELECTED WATER QUALITY DATA
LAMEFOOT MINE INFILTRATION POND (LF-I)

Sample ID: LF-I	Field Parameters			Laboratory Analytical Parameters								
	Field pH (units)	Field SC (µmhos/cm)	Field Temp. °C	Lab pH (units)	Lab SC (µmhos/cm)	As-T (mg/L)	As-D (mg/L)	Cu-T mg/L)	Cu-D (mg/L)	Fe-T (mg/L)	Fe-D (mg/L)	Pb-T (mg/L)
State GW Criteria:	6.5-8.5					0.05		1.0		0.30		0.05
4/6/93	8.2	1572	6.0	7.9	1430	< 0.04		0.011		3.17		< 0.04
5/11/93	8.1	1420	9.0	7.56	1330			0.035				< 0.04
6/8/93	7.7	1651	20	7.17				0.014				< 0.04
7/7/93	7.9	1450	19	7.29	1400				0.008			
8/5/93	7.8	1510	23	6.83	1520			0.019				< 0.04
9/15/93	8.1	1546	16					0.02				< 0.02
10/6/93	8.2	1542	13					0.01				< 0.02
11/2/93	7.9	2310	5.0	8.3	2150				< 0.01			
4/6/94	7.9	1313	10					0.05				< 0.02
5/10/94	8.0	1111	12					0.04				< 0.02
Summary Statistics:												
LF-1 Maximum	8.2	2310	23	8.3	2150	0.04		0.05	0.01	3.17		0.04
LF-1 Minimum	7.7	1111	5	6.83	1330	0.04		0.01	0.008	3.17		0.02
LF-1 Average	8.0	1543	13.3	7.5	1566	0.04		0.02	0.009	3.17		0.03
n	10	10	10	6	5	1		8	2	1		8
≥State GW Criteria?	NO					NO		NO		YES		NO

(1) Total $NO_3 + NO_2$ - N

(2) Dissolved NO_3-N

Notes: LF-I dry or frozen and not sampled: Dec. 93, Jan. 94, Feb. 94, Mar. 94

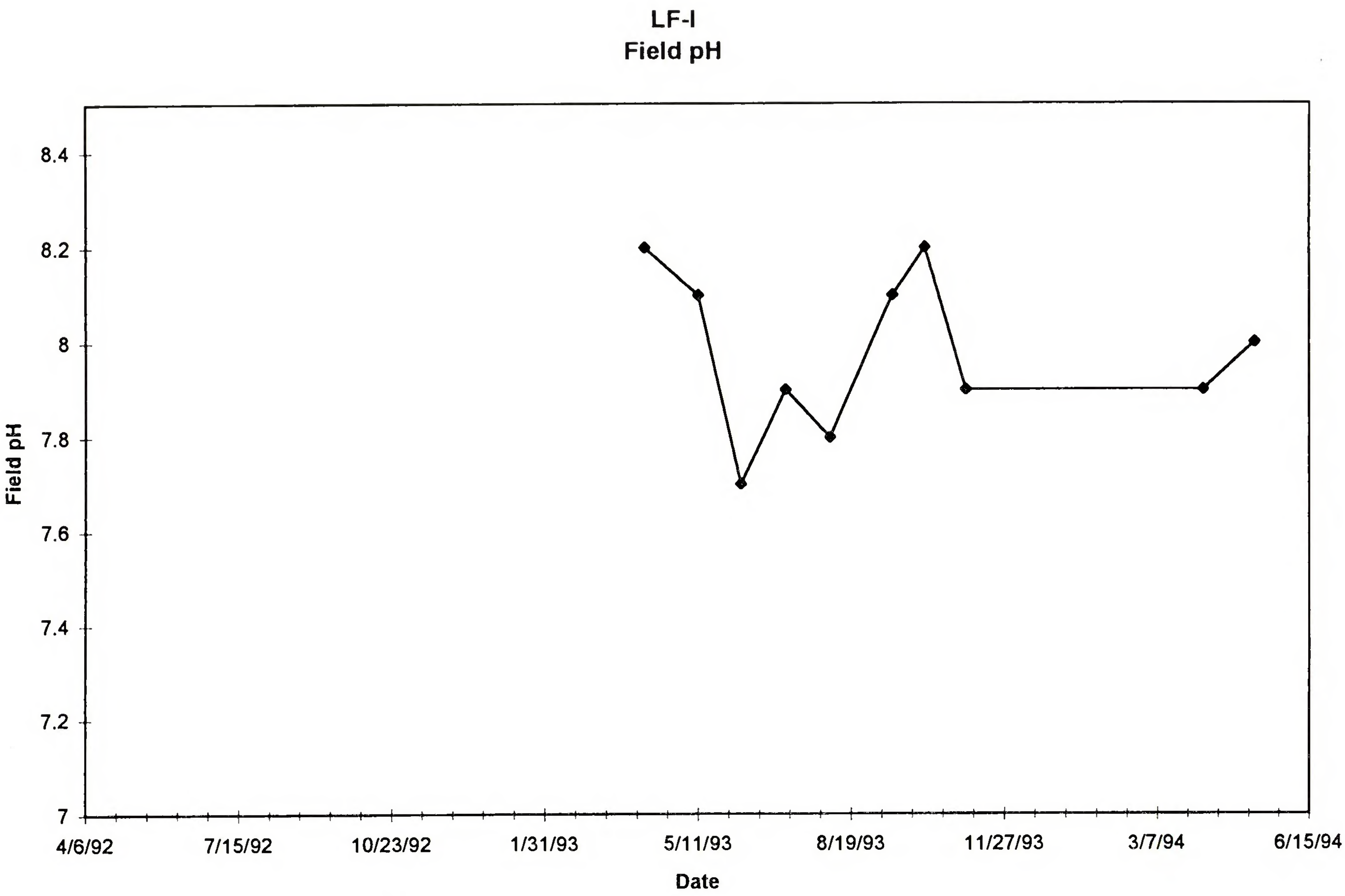
SUMMARY OF SELECTED WATER QUALITY DATA
LAMEFOOT MINE INFILTRATION POND (LF-I)

Sample ID: LF-1	Laboratory Analytical Parameters									
	Pb-D (mg/L)	Mn-T (mg/L)	Mn-D (mg/L)	Zn-T (mg/L)	Zn-D (mg/L)	Total Alk. As $CaCO_3$ (mg/L)	Sulfate (mg/L)	NO_3 As N (mg/L)	TDS (mg/L)	TSS (mg/L)
State GW Criteria:		0.05		5.0			250	10	500	
4/6/93		0.112		0.038				70.5		91.4
5/11/93				0.165				33.3		358
6/8/93				0.035				61.7 (1)		182
7/7/93	< 0.04				0.033			54 (1)		17.9
8/5/93				0.075				1.63		15.6
9/15/93				0.05				64.4 (2)		18
10/6/93				0.06				59.8 (2)		82
11/2/93	< 0.02		0.05		0.03	208	545	136 (2)	978	< 2
4/6/94				0.05				24.04 (2)		
5/10/94				0.04				11.74 (2)		
Summary Statistics:										
LF-1 Maximum	0.04	0.112	0.05	0.165	0.033	208	545	136	978	358
LF-1 Minimum	0.02	0.112	0.05	0.035	0.030	208	545	1.63	978	2
LF-I Average	0.03	0.112	0.05	0.064	0.032	208	545	51.7	978	95.9
n	2	1	1	8	2	1	1	10	1	8
≥State GW Criteria?		YES		NO			YES	YES	YES	

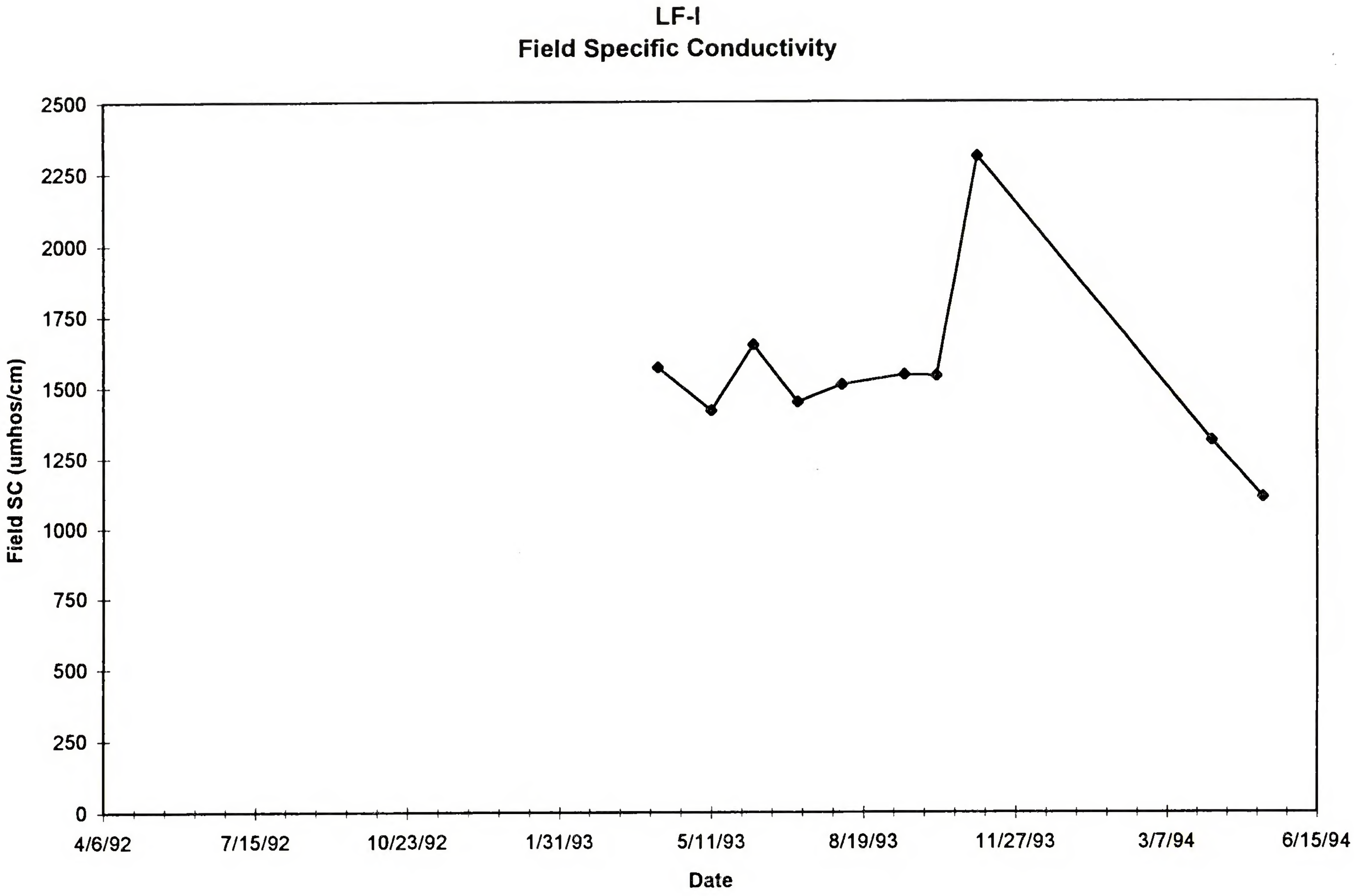
(1) Total $NO_3 + NO_2$ - N

(2) Dissolved NO_3-N

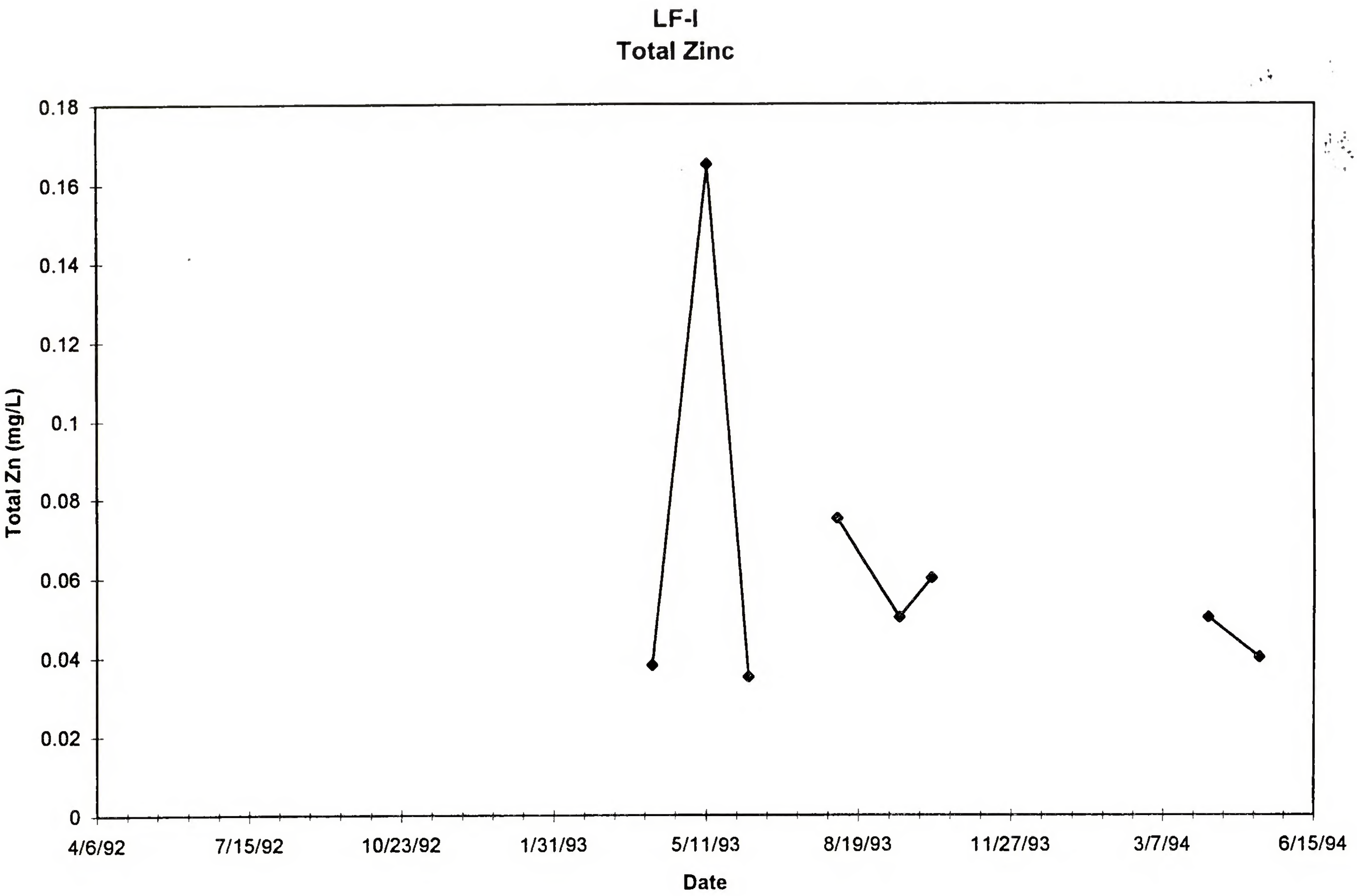
Notes: LF-1 dry or frozen and not sampled: Dec. 93, Jan. 94, Feb. 94, Mar. 94


LF-I
Field pH
Field pH
8.4
8.2
8
7.8
7.6
7.4
7.2
7
4/6/92
7/15/92
10/23/92
1/31/93
5/11/93
8/19/93
11/27/93
3/7/94
6/15/94
Date

LFIpH.xls

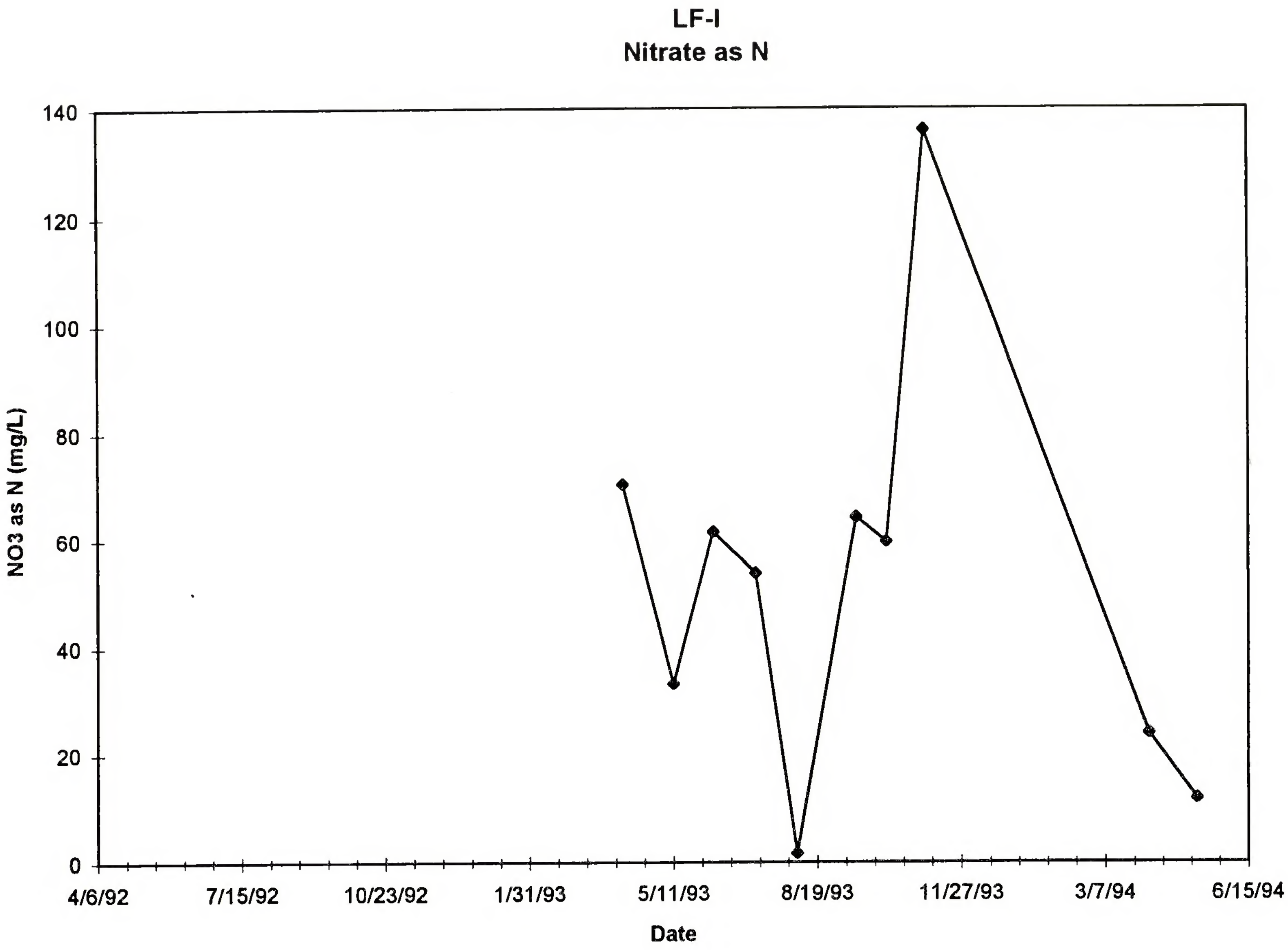

LF-I
Field Specific Conductivity
2500
2250
2000
1750
1500
1250
1000
750
500
250
0
Field SC (umhos/cm)
4/6/92
7/15/92
10/23/92
1/31/93
5/11/93
8/19/93
11/27/93
3/7/94
6/15/94
Date

LFISC.xls



LF-I
Total Zinc

LFITOTZn.xls


LF-I
Nitrate as N
NO3 as N (mg/L)
140
120
100
80
60
40
20
0
4/6/92
7/15/92
10/23/92
1/31/93
5/11/93
8/19/93
11/27/93
3/7/94
6/15/94
Date

LFINO3.xls

SUMMARY OF SELECTED WATER QUALITY DATA
LAMEFOOT MINE MONITORING WELL LF-1

Sample ID: LF-1	Field Parameters			Laboratory Analytical Parameters								
	Field pH (units)	Field SC (µmhos/cm)	Field Temp. °C	Lab pH (units)	Lab SC (µmhos/cm)	As-T (mg/L)	As-D (mg/L)	Cu-T mg/L)	Cu-D (mg/L)	Fe-T (mg/L)	Fe-D (mg/L)	Pb-T (mg/L)
State GW Criteria:	6.5-8.5					0.05		1.0		0.30		0.05
4/7/92	7.9	921	5.0	7.72	806		0.026		< 0.005		0.04	
5/5/92	7.8	906	5.0	8.02	769		0.047		< 0.005		0.09	
6/9/92	7.7	892	6.0	7.76	798		< 0.04		0.006		1.48	
7/8/92	7.8	902	6.0	8.08	793		< 0.002		0.056		0.12	
10/14/92	7.7	910	5.0	8	865		0.065		0.076		22.8	
11/10/92	7.8	906	4.0	7.8	826		< 0.04		0.005		0.584	
12/2/92	7.8	946	4.0	8.07	897		< 0.05		< 0.004		0.056	
1/5/93	7.3	902	4.0	8	862		0.04		0.004		0.124	
2/3/93	7.1	1005	6.0	7.8	851		< 0.04		< 0.003		< 0.009	
3/2/93	7.4	945	6.3	7.84	810		< 0.04		< 0.003		< 0.009	
4/6/93	8.2	759	7.0	7.9	717		< 0.04		< 0.004		0.018	
5/11/93	7.8	960	10	7.72	765		< 0.04		< 0.004		< 0.009	
6/8/93	7.7	831	9.0	7.98	800		< 0.04		< 0.004		< 0.009	
7/7/93	7.6	838	10	7.57	779		< 0.04		< 0.001		< 0.02	
8/5/93	7.7	740	12	7.64	772		< 0.04		0.002		< 0.02	
9/14/93	8.1	788	10	8.1	769		0.008		< 0.01			
10/6/93	7.8	835	10	7.3	755		0.007		< 0.01			
11/2/93	7.7	852	9.0	8.1	782		0.008		< 0.01			
12/7/93	7.0	871	7.8	8.2	784		0.007		< 0.01			
2/8/94	7.7	780	4.0	7.7	800		0.006		< 0.05			
3/8/94	7.7	857	9.0	8.1	791		0.007		< 0.01			
4/6/94	7.4	781	10	7.8	766		0.006		< 0.01			
5/10/94	7.5	798	8.0	7.8	782		0.006		< 0.01			
Summary Statistics:												
LF-1 Maximum	8.2	1005	12	8.2	897		0.065		0.076		22.8	
LF-1 Minimum	7	740	4	7.3	717		0.002		0.001		0.009	
LF-1 Average	7.7	866	7.3	7.9	797		0.028		0.013		1.693	
n	23	23	23	23	23		23		23		15	
≥State GW Criteria?	NO						YES (3)		NO (3)		YES (3)	

(1) Total $NO_3 + NO_2$ - N

(2) Dissolved NO_3 - N

(3) Dissolved concentration compared to GW criteria for total metals

23295/LF-1.XLS 7/14/94(2:01 PM)/RPT/5

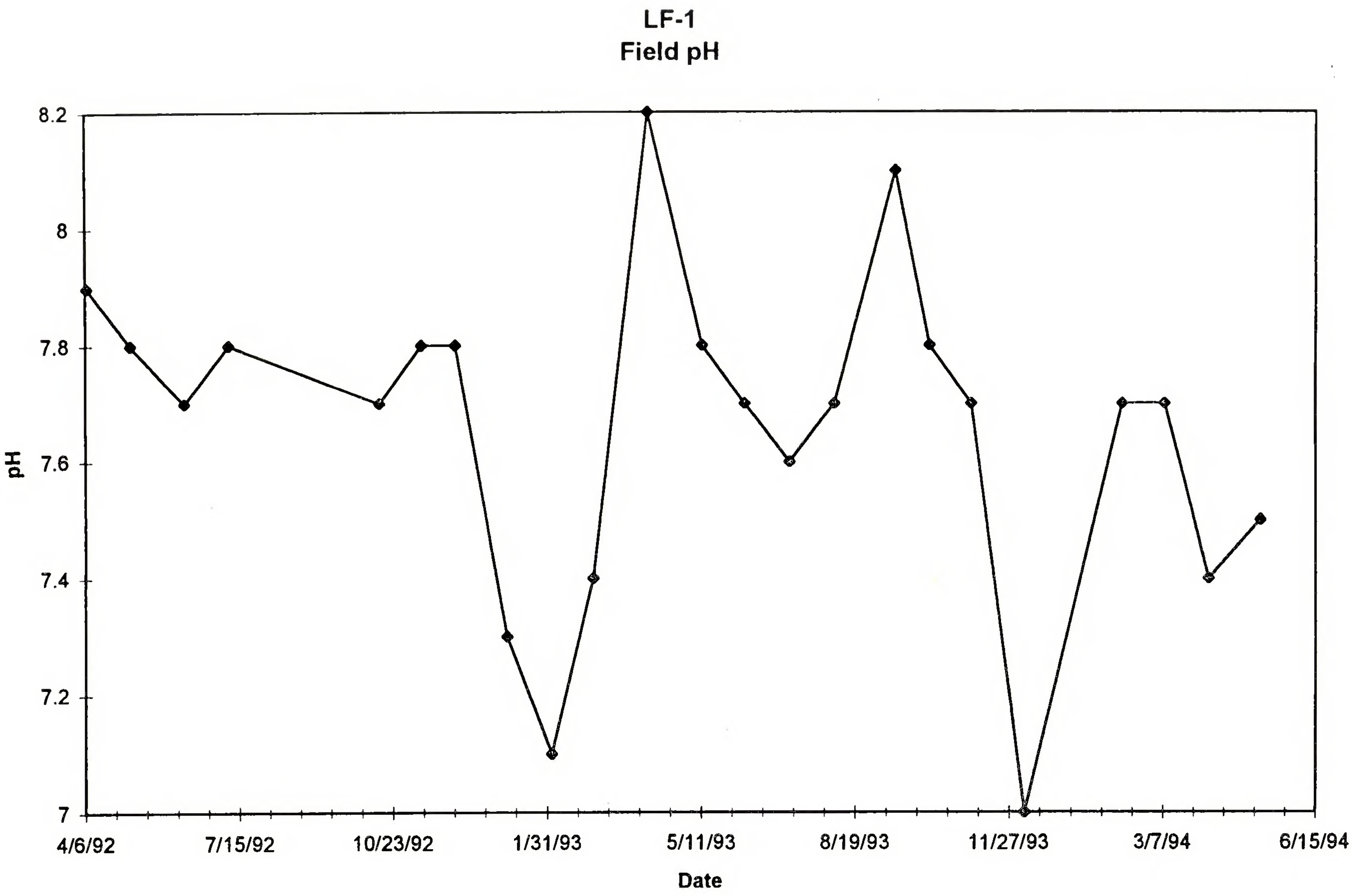
SUMMARY OF SELECTED WATER QUALITY DATA
LAMEFOOT MINE MONITORING WELL LF-1

Sample ID: LF-1	Laboratory Analytical Parameters									
	Pb-D (mg/L)	Mn-T (mg/L)	Mn-D (mg/L)	Zn-T (mg/L)	Zn-D (mg/L)	Total Alk. as $CaCO_3$ (mg/L)	Sulfate (mg/L)	NO_3 As N (mg/L)	TDS (mg/L)	TSS (mg/L)
State GW Criteria:		0.05		5.0			250	10	500	
4/7/92	< 0.001		< 0.001		0.006	254	203	0.59 (1)	552	863
5/5/92	0.019		0.004		< 0.005	247	214	0.99	544	50.9
6/9/92	< 0.04		0.028		0.021	234	211	0.87 (1)	616	292.5
7/8/92	< 0.001		0.007		0.016	246	189	0.74	521	53
10/14/92	< 0.049		0.458		0.13	274	218.3	0.66 (1)	610	434.7
11/10/92	0.1		0.027		0.014	266	224	0.66 (1)	629	1737.3
12/2/92	< 0.05		0.006		0.003	281	226	0.69 (1)	572	310
1/5/93	0.08		< 0.002		0.009	282	218	1.99 (1)	571	453
2/3/93	< 0.03		< 0.002		0.021	271	221	0.58 (1)	619	357
3/2/93	< 0.03		< 0.002		0.015	270	204	0.93 (1)	578	327
4/6/93	< 0.04		< 0.001		0.003	256	154	1.28 (1)	474	111
5/11/93	< 0.04		< 0.001		0.003	240	181	1.27 (1)	455	58.6
6/8/93	< 0.04		< 0.001		< 0.001	246	188	0.93 (1)	531	186
7/7/93	< 0.04		< 0.002		< 0.002	255	192	0.89 (1)	547	34.2
8/5/93	< 0.04		< 0.002		< 0.002	261	181	0.86 (1)	553	63
9/14/93	< 0.02		0.07		< 0.01	264	179	0.87 (2)	510	160
10/6/93	< 0.02		< 0.01		0.02	282	177	0.77 (2)	526	26
11/2/93	< 0.02		< 0.01		< 0.01	270	200	0.7 (2)	554	8
12/7/93	< 0.02		< 0.01		< 0.01	262	179	0.68 (2)	490	< 2
2/8/94	< 0.1		< 0.05		< 0.05	284	173	0.69 (2)	542	< 2
3/8/94	< 0.02		< 0.01		< 0.02	274	163	0.88 (2)	520	2
4/6/94	< 0.02		< 0.01		< 0.01	258	152	1.07 (2)	478	10
5/10/94	< 0.02		< 0.01		< 0.01	278	187	1.11 (2)	492	< 2
Summary Statistics:										
LF-1 Maximum	0.1		0.458		0.13	284	226	1.99	629	1737.3
LF-1 Minimum	0.001		0.001		0.001	234	152	0.58	455	2
LF-1 Average	0.037		0.031		0.017	263	193	0.90	543	241
	23		23		23	23	23	23	23	23
≥State GW Criteria?	YES (3)		YES (3)		NO (3)		YES	YES	YES	

(1) $NO_3 + NO_2$ - N

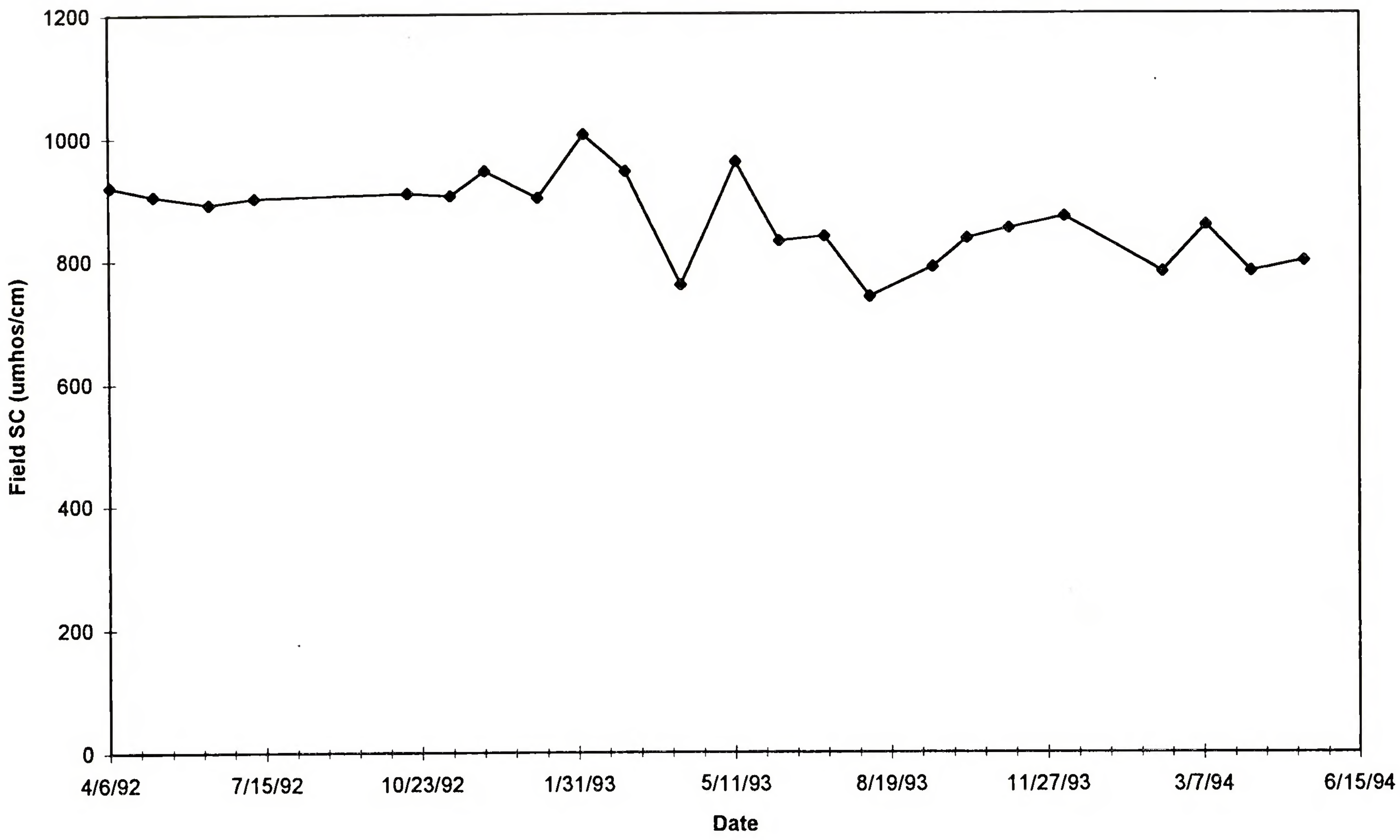
(2) Dissolved NO_3 - N

(3) Dissolved concentration compared to GW criteria for total metals

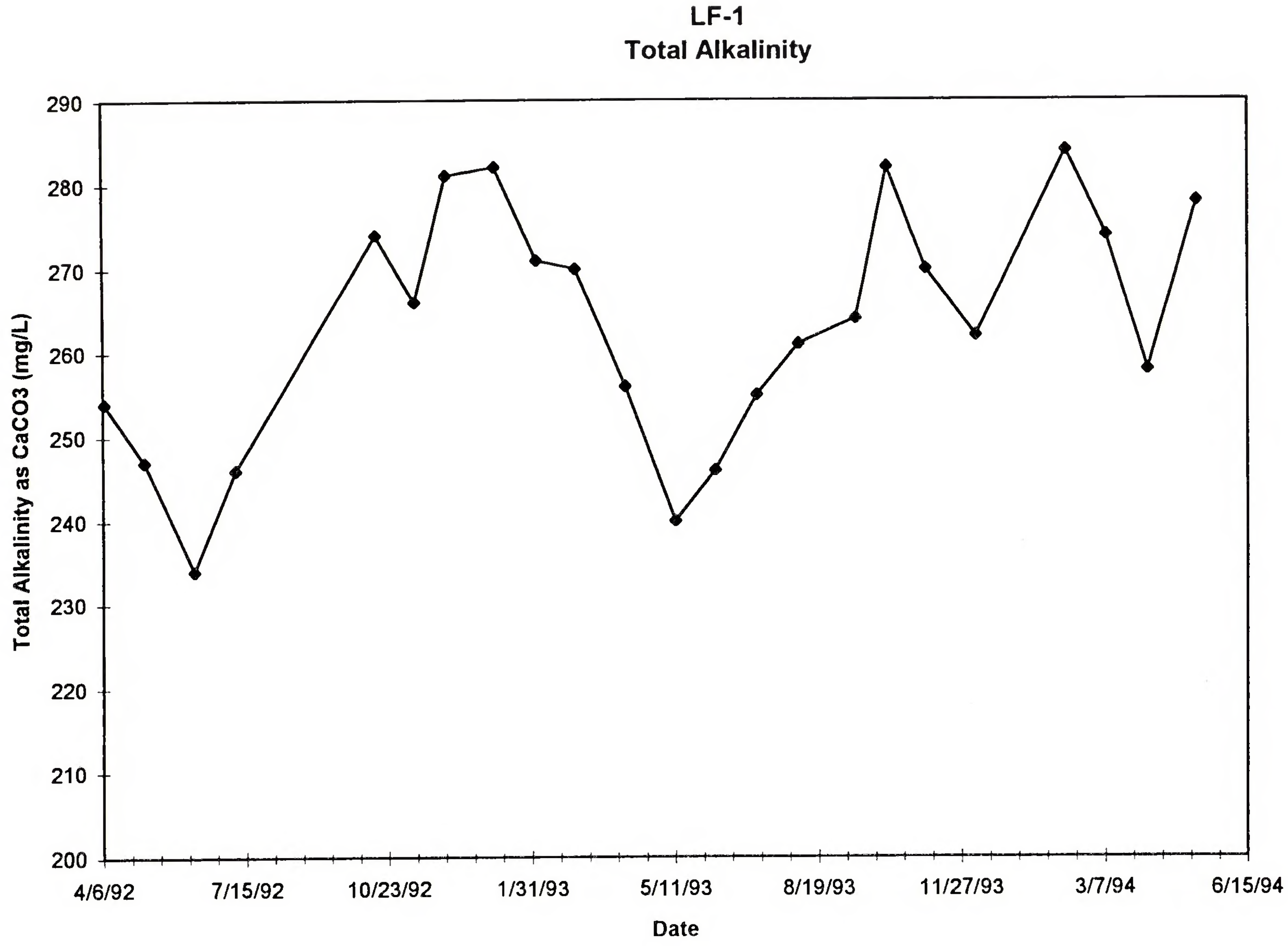

LF-1
Field pH
8.2
8
7.8
7.6
7.4
7.2
7
pH
4/6/92
7/15/92
10/23/92
1/31/93
5/11/93
8/19/93
11/27/93
3/7/94
6/15/94
Date

LF1pH.xls

LF-1
Field Specific Conductivity

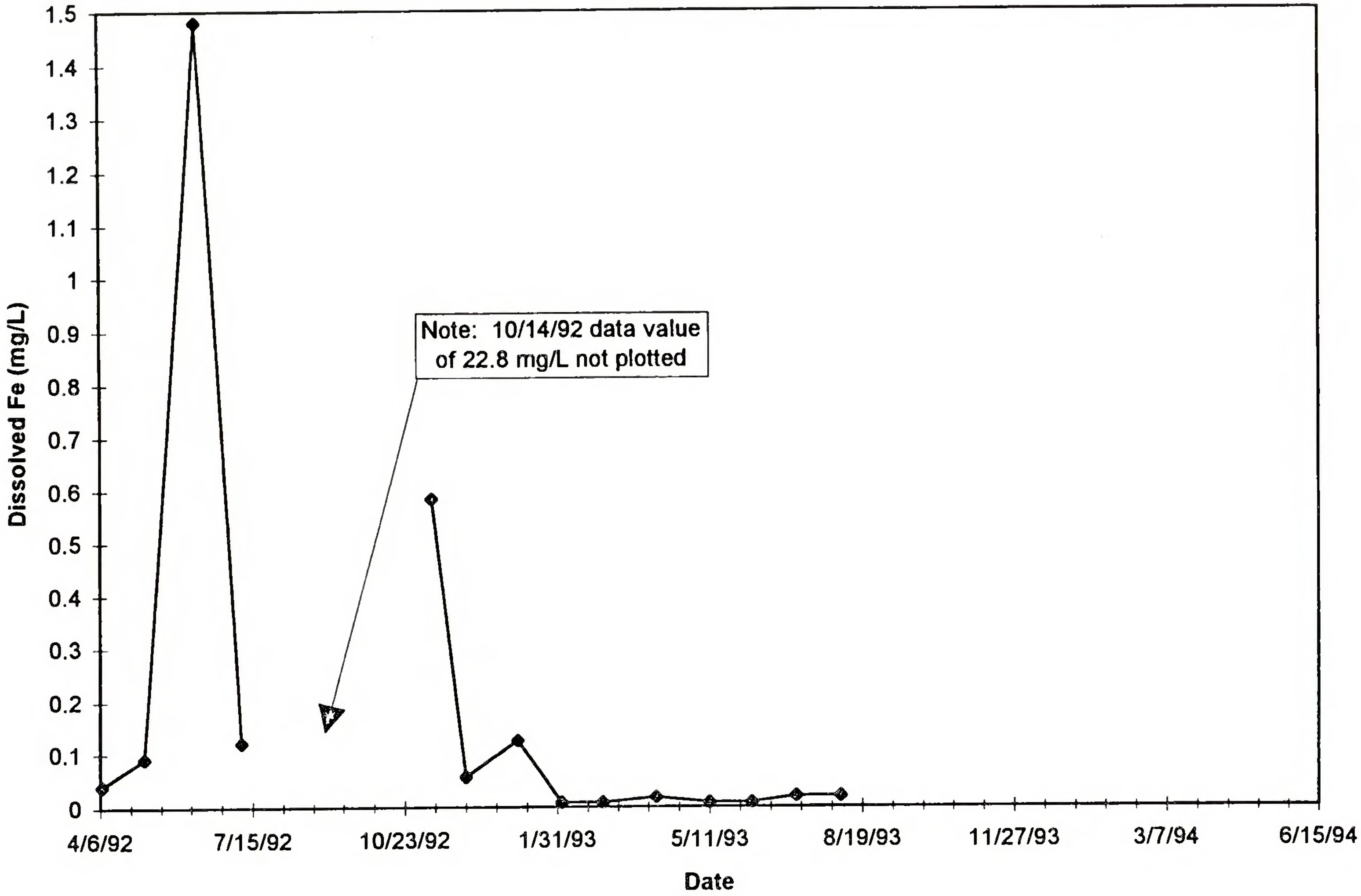


LF1SC.xls

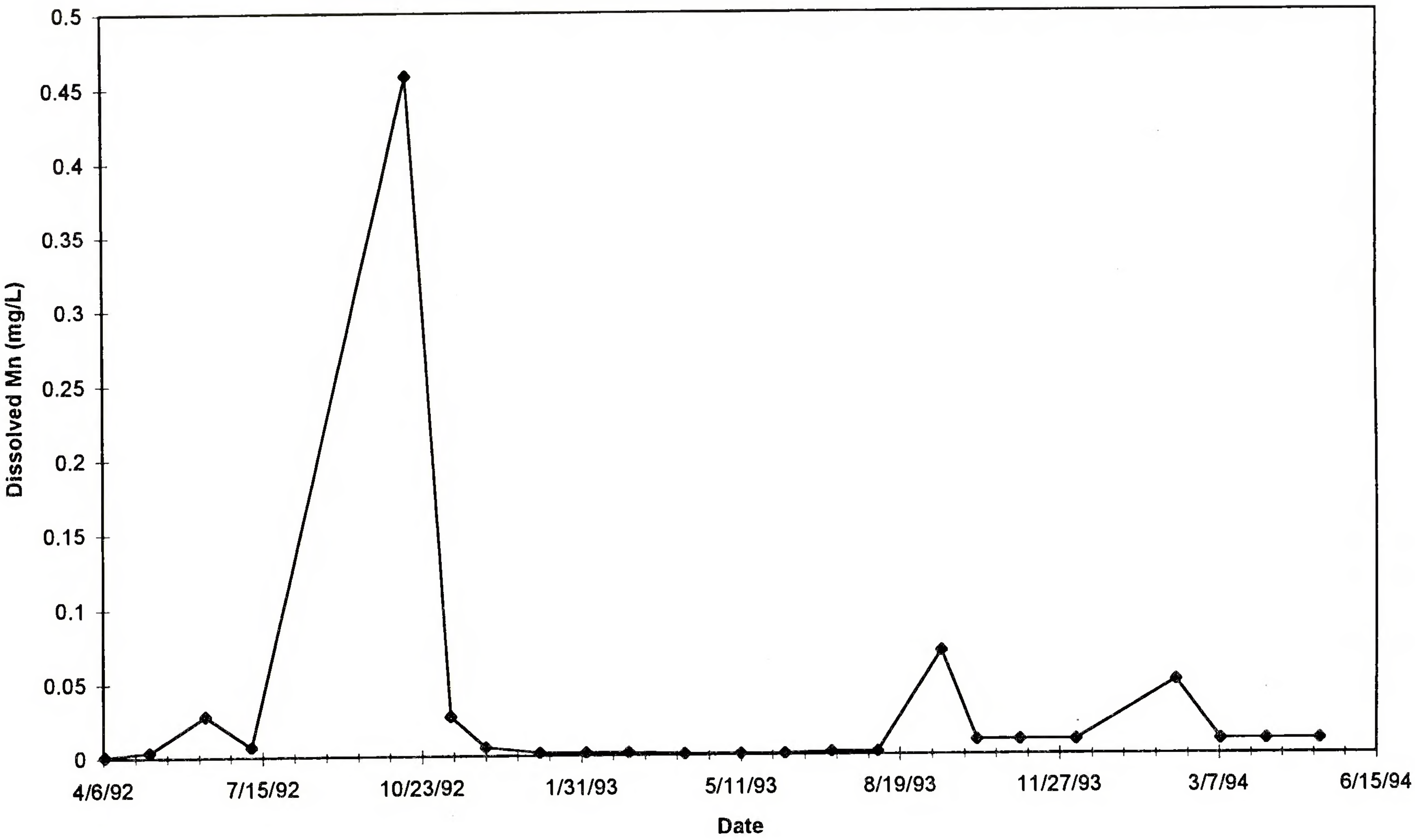


LF1TotAlk.xls

LF-1
Dissolved Iron

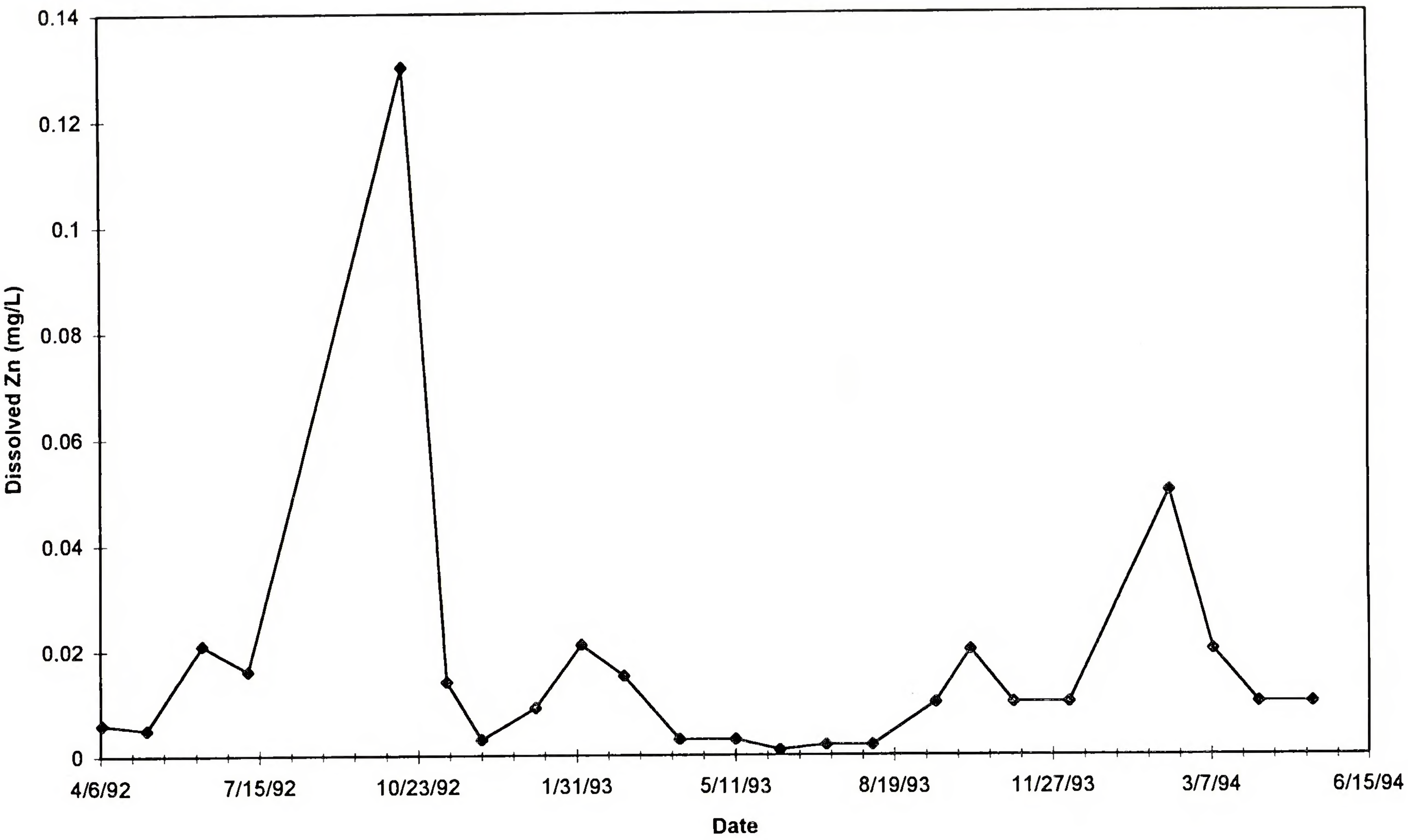

Note: 10/14/92 data value of 22.8 mg/L not plotted

LF1DisFe.xls

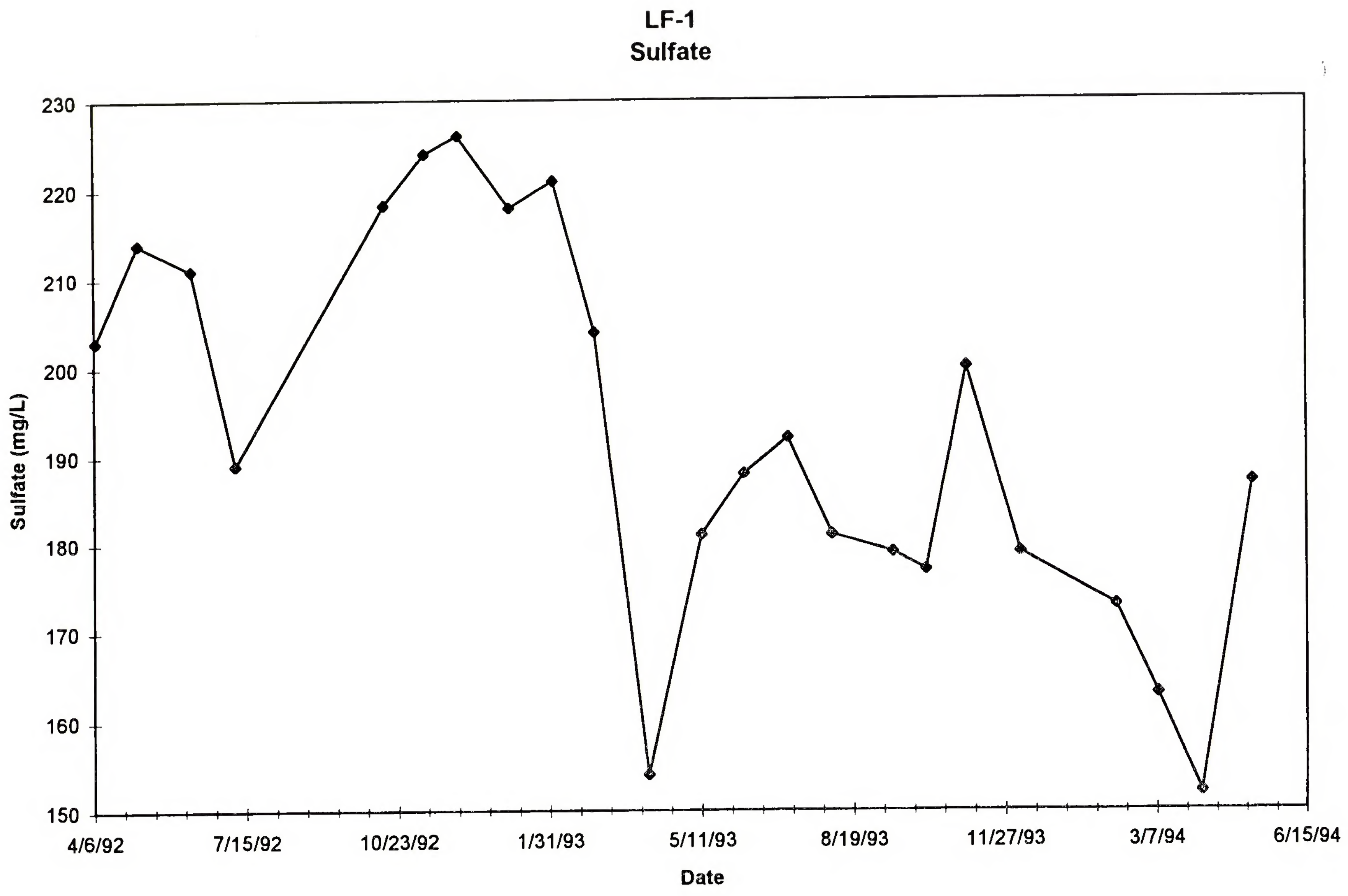


LF-1
Dissolved Manganese

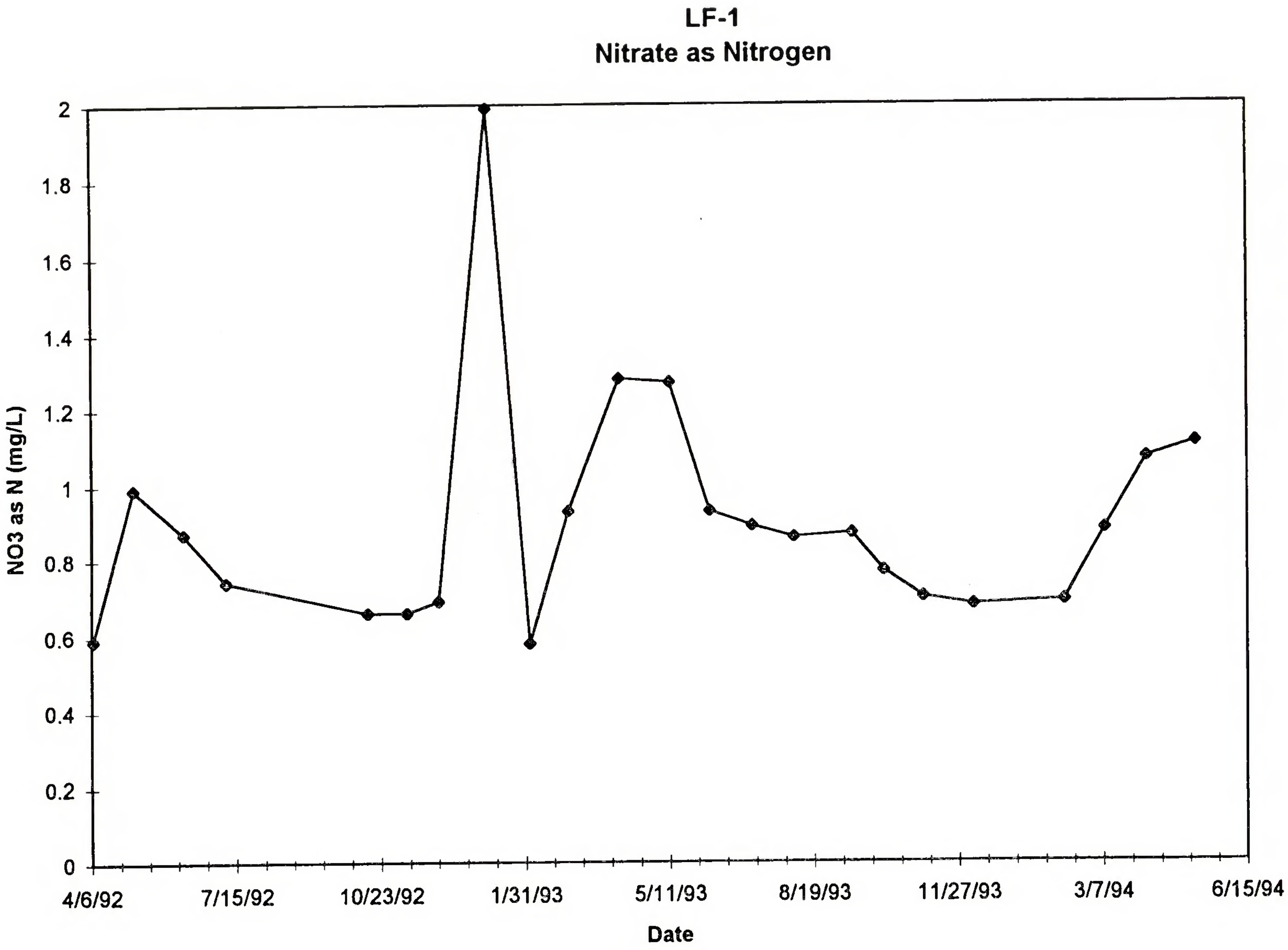
LF-1
Dissolved Zinc



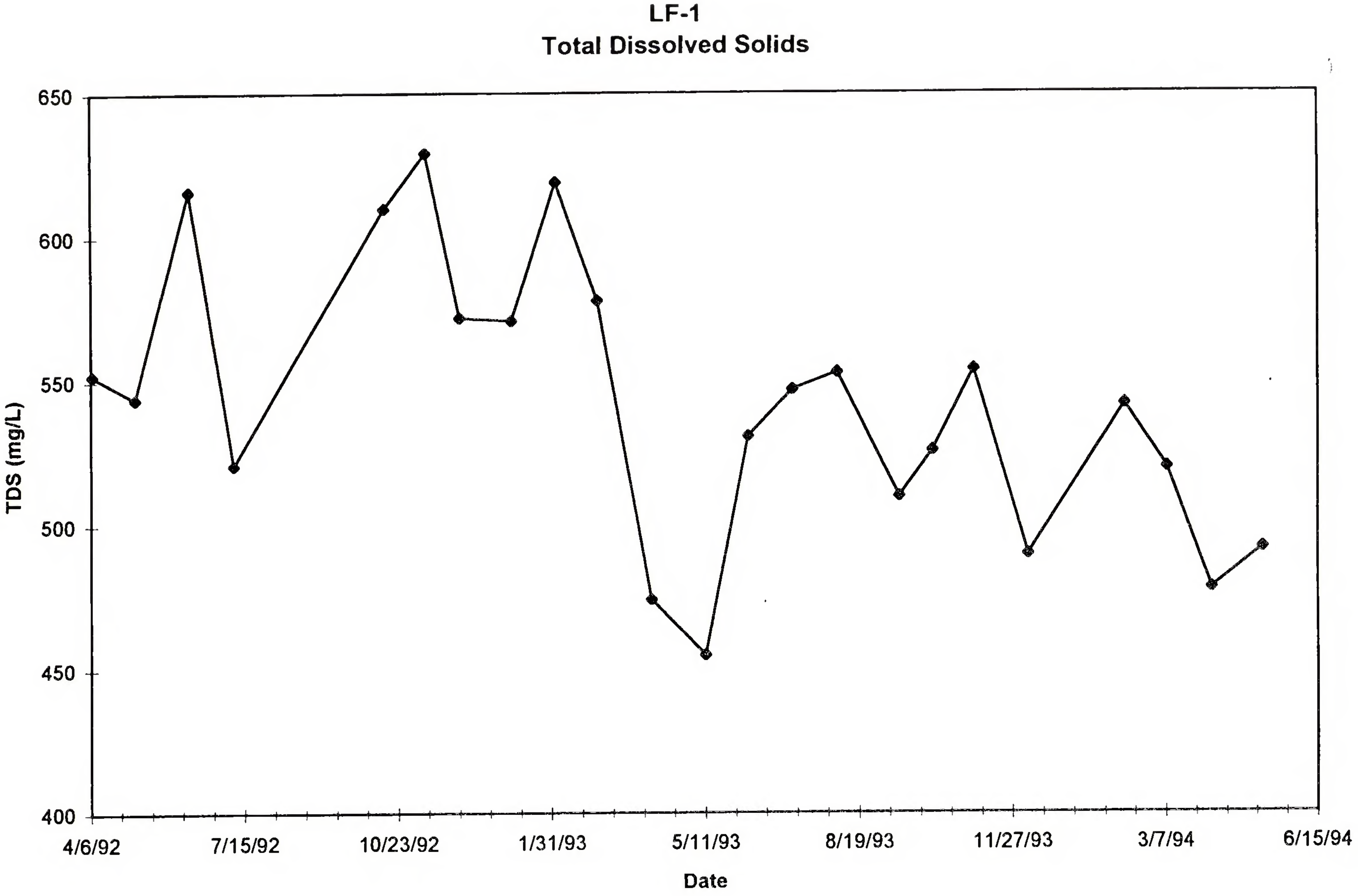
LF1DisZn.xls


LF-1
Sulfate
Sulfate (mg/L)
230
220
210
200
190
180
170
160
150
4/6/92
7/15/92
10/23/92
1/31/93
5/11/93
8/19/93
11/27/93
3/7/94
6/15/94
Date

LF1Slf.xls


LF-1
Nitrate as Nitrogen
NO3 as N (mg/L)
2
1.8
1.6
1.4
1.2
1
0.8
0.6
0.4
0.2
0
4/6/92
7/15/92
10/23/92
1/31/93
5/11/93
8/19/93
11/27/93
3/7/94
6/15/94
Date

LF1NO3.xls


LF-1
Total Dissolved Solids
650
600
550
500
450
400
TDS (mg/L)
4/6/92
7/15/92
10/23/92
1/31/93
5/11/93
8/19/93
11/27/93
3/7/94
6/15/94
Date

LF1TDS.xls

SUMMARY OF SELECTED WATER QUALITY DATA
LAMEFOOT MINE MONITORING WELL LF-2

Sample ID:	Field Parameters			Laboratory Analytical Parameters								
LF-2	Field pH (units)	Field SC (µmhos/cm)	Field Temp. °C	Lab pH (units)	Lab SC (µmhos/cm)	As-T (mg/L)	As-D (mg/L)	Cu-T mg/L)	Cu-D (mg/L)	Fe-T (mg/L)	Fe-D (mg/L)	Pb-T (mg/L)
State GW Criteria:	6.5-8.5					0.05		1.0		0.30		0.05
4/7/92	7.8	1170	5.0	7.89	1078		< 0.002		0.009		0.04	
5/5/92	7.7	1100	5.0	7.89	955		0.033		0.007		0.6	
6/9/92	7.4	1040	6.0	7.65	904		< 0.04		0.006		0.43	
7/8/92	7.5	1110	6.0	8.0	994		< 0.002		0.007		0.16	
8/5/92	7.6	1096	6.0	7.85	955		< 0.035		0.005		0.65	
9/8/92	7.5	1020	7.0	7.88	1020		0.035		< 0.005		0.04	
10/14/92	7.6	1000	5.0	8.0	976		0.108		0.213		48.4	
11/10/92	7.7	1040	5.0	7.8	956		< 0.04		0.005		0.183	
12/2/92	7.5	1100	5.0	7.8	1010		< 0.05		< 0.004		0.083	
1/5/93	7.3	1105	4.0	7.7	1030		< 0.04		0.004		0.088	
2/3/93	7.1	1180	6.0	7.7	1010		< 0.04		< 0.003		< 0.009	
3/2/93	7.4	1195	7.0	7.78	1028		< 0.04		0.004		< 0.009	
4/6/93	8.2	1106	7.9	7.96	1050		< 0.04		< 0.004		0.022	
5/11/93	7.6	1180	10	7.7	848		< 0.04		< 0.004		< 0.009	
6/8/93	7.7	795	9.0	7.9	749		< 0.04		0.006		< 0.009	
7/7/93	7.6	838	10	7.53	762		< 0.04		0.004		< 0.02	
8/5/93	7.7	646	11	7.19	807		< 0.04		0.004		< 0.02	
9/14/93	8.1	805	9.0	8.1	822		0.006		0.01			
10/6/93	7.7	967	9.0	7.6	877		0.005		< 0.01			
11/2/93	7.6	1042	8.0	8.0	968		0.005		< 0.01			
12/7/93	7	956	7.7	8.2	873		0.005		< 0.01			
2/8/94	7.6	881	5.0	7.6	900		0.007		< 0.01			
3/8/94	7.8	1038	8.0	8.0	967		0.006		< 0.01			
4/6/94	7.4	989	10	7.7	967		0.004		< 0.01			
5/10/94	7.4	767	8.0	7.6	770		0.004		< 0.01			
Summary Statistics:												
LF-2 Maximum	8.2	1195	11	8.2	1078		0.108		0.213		48.4	
LF-2 Minimum	7	646	4	7.19	749		0.002		0.003		0.009	
LF-2 Average	7.6	1007	7.2	7.8	931		0.028		0.015		2.987	
n	25	25	25	25	25		25		25		17	
≥State GW Criteria?	NO						YES (3)		NO (3)		YES (3)	

(1) Total $NO_3 + NO_2$ - N

(2) Dissolved NO_3-N

(3) Dissolved concentration compared to GW criteria for total metals

Notes: Well inaccessible and not sampled: Jan. 94

SUMMARY OF SELECTED WATER QUALITY DATA
LAMEFOOT MINE MONITORING WELL LF-2

Sample ID:	Laboratory Analytical Parameters									
LF-2	Pb-D (mg/L)	Mn-T (mg/L)	Mn-D (mg/L)	Zn-T (mg/L)	Zn-D (mg/L)	Total Alk. As $CaCO_2$ (mg/L)	Sulfate (mg/L)	NO_3 As N (mg/L)	TDS (mg/L)	TSS (mg/L)
State GW Criteria:		0.05		5.0			250		500	
4/7/92	< 0.001		0.002		0.025	292	335	0.52 (1)	799	103.2
5/5/92	0.002		0.021		0.013	264	319	0.55 (1)	776	1658
6/9/92	< 0.04		0.006		0.01	270	267	0.47 (1)	691	1750
7/8/92	< 0.001		0.009		0.019	287	286	0.48 (1)	699	585.6
8/5/92	< 0.041		0.012		0.011	259		0.4 (1)	1414	3096
9/8/92	0.012		0.002		0.006	265	326	0.41 (1)	746	930
10/14/92	< 0.049		0.676		0.257	300	265.2	0.5 (1)	716	473.3
11/10/92	< 0.04		0.007		0.016	285	275	0.44 (1)	715	638.3
12/2/92	< 0.05		0.005		0.009	301	287	0.49 (1)	875	1830
1/5/93	< 0.03		< 0.002		0.012	303	297	1.03 (1)	686	1950
2/3/93	< 0.03		< 0.002		0.007	300	289	0.43 (1)	728	3110
3/2/93	< 0.03		< 0.002		0.011	302	321	0.68 (1)	754	1560
4/6/93	< 0.04		< 0.001		0.013	297	327	0.92 (1)	782	23.4
5/11/93	< 0.04		< 0.001		0.006	250	227	0.76 (1)	513	73.3
6/8/93	< 0.04		< 0.001		< 0.001	251	172	0.5 (1)	538	316
7/7/93	< 0.04		< 0.002		0.006	265	163	0.42 (1)	503	16.5
8/5/93			< 0.002		0.004	274	185	0.43 (1)	586	33.5
9/14/93	< 0.02		0.05		< 0.01	240	290	0.81 (2)	616	780
10/6/93	< 0.02		< 0.01		< 0.01	302	263	1.01 (2)	666	< 2
11/2/93	< 0.02		< 0.01		< 0.01	296	278	1.01 (2)	710	< 2
12/7/93	< 0.02		< 0.01		< 0.01	282	218	1.26 (2)	602	16
2/8/94	< 0.02		0.02		0.03	312	234	0.73 (2)	642	78
3/8/94						306	249	0.87 (2)	648	2
4/6/94	< 0.02		< 0.01		< 0.01	284	247	1.97 (2)	670	< 2
5/10/94	< 0.02		< 0.01		< 0.01	270	187	0.7 (2)	484	< 2
Summary Statistics:										
LF-2 Maximum	0.05		0.676		0.257	312	335	1.97	1414	3110
LF-2 Minimum	0.001		0.001		0.001	240	163	0.4	484	2
LF-2 Average	0.027		0.036		0.022	282	263	0.7	702	761.2
n	23		24		24	25	24	25	25	25
≥State GW Criteria?	YES (3)		YES (3)		NO (3)		YES	YES	YES	

(1) Total $NO_3 + NO_2$ - N

(2) Dissolved NO_3-N

(3) Dissolved concentration compared to GW criteria for total metals

Notes: Well inaccessible and not sampled: Jan. 94

NOTES ON TABLES OF SELECTED WQ DATA FOR LF-I, LF-1, AND LF-2 LAMEFOOT MINE

Tables are constructed in Excel 5.0 with appropriate formatting in cells to calculate maxima, minima, averages, and whether parameter equals or exceeds state groundwater criteria. When entering new data, care will need to be taken that cells for new data are formatted to calculate in updated statistics. Graphs should be updated to include new data in the date and concentration ranges.

Abbreviations include the following:

n = number of observations
SC = specific conductivity
T = total recoverable (unfiltered)
D = dissolved (filtered)
As = arsenic
Cu = copper
Fe = iron
Pb = lead
Mn = manganese
Zn = zinc
Alk. = alkalinity (as $CaCO_3$)
NO_3 = nitrate (as N)
TDS = total dissolved solids
TSS = total suspended solids

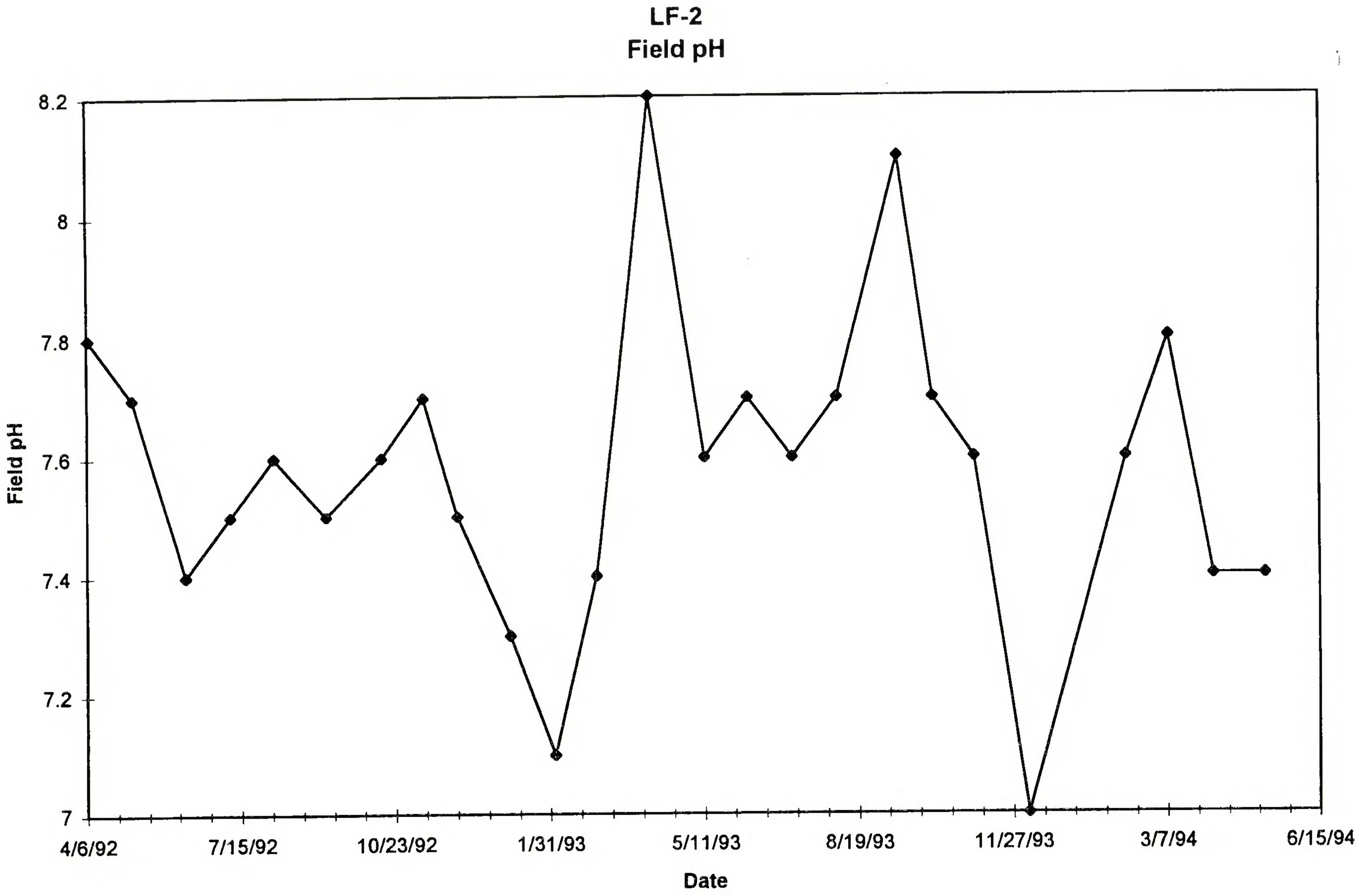
Graphs of selected indicator parameters (field pH and specific conductivity, total alkalinity, iron, manganese, zinc, sulfate, nitrate, and TDS) have been constructed and linked to the Excel spreadsheets. Parameters have been chosen based on key indicators of potential acid rock drainage and parameters on the short list to be analyzed most frequently in future monitoring. Graphs were constructed for an indicator parameter when there were more than two observations for that parameter at that site.

Where qualified data (such as concentrations below the detection limit [i.e., < 0.05 mg/L]) exist, the parameter has been assumed to be present at the detection limit for purposes of calculating maxima, minima, and averages. The "YES/NO" in the last row indicates that the maximum value for that parameter equals or exceeds the state groundwater criterion. State groundwater criteria are stated in terms of total metals concentrations. However, it is assumed that if a dissolved (filtered) sample equals or exceeds the criteria, a corresponding total sample also would equal or exceed the criteria, since total concentrations generally exceed dissolved concentrations when measured on aliquots of the same sample. Columns for chemical parameters are divided into a column preceding the number (for < or other appropriate symbols), the column for the chemical result to be used in calculations, and a column following the number (for footnotes).

Selected data results appeared to be lab errors, in that results were clearly anomalous and appeared unreasonable. These results were not entered in the database tables, including the following: The conductance result for the 6/8/93 sample for LF-I (the lab reported 0.2 umhos/cm, anomalously low); the sulfate result for the 8/5/92 sample for LF-2 (the lab reported 0.3 mg/L, anomalously low); and the lead result for the 8/5/93 sample for LF-2 (the lab reported an anomalously high value which repeated the result for dissolved magnesium). Anomalously high results were reported for dissolved iron for the 10/14/92 samples from LF-1 and LF-2 (22.8 and 48.4 mg/L, respectively, for LF-1 and LF-2). Although these dissolved iron data are suspect, it is not clear without more information that they can be attributed to lab error. Thus, these dissolved iron results have been noted as anomalous but retained in the database.

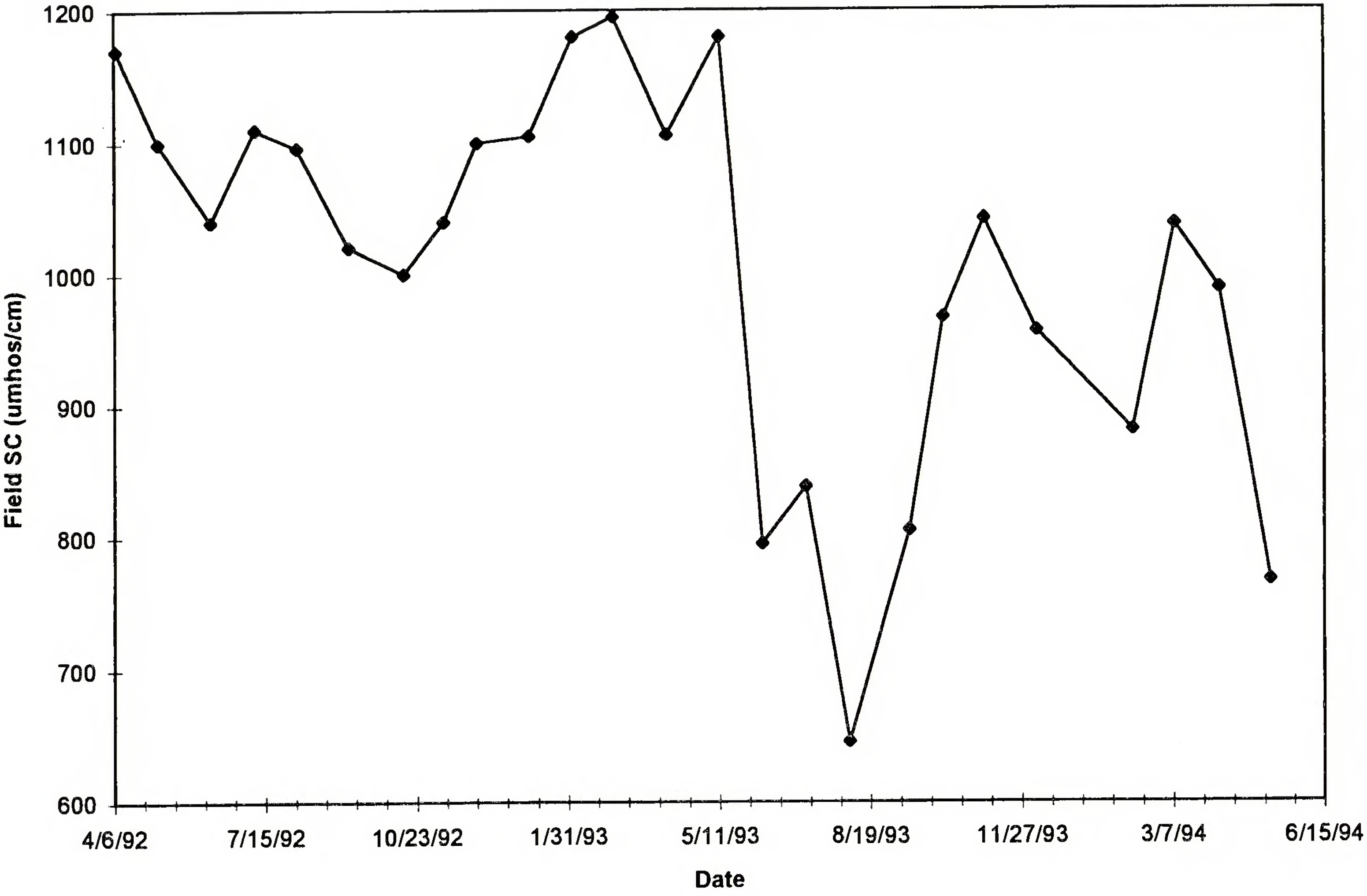
Cyanide is not included as an indicator parameter and historically has not been detected in the infiltration pond LF-I and wells LF-1 and LF-2, except for results reported from monthly sampling of LF-1 and LF-2 in September and October 1993. Results from those two months, reported during a changeover from SVL to ACZ labs, indicated total cyanide ranging from 0.008 to 0.003 mg/L. These values are well below the previous laboratory reporting limits of 0.01 mg/L for WAD (weak acid dissociable) cyanide, which was not detected by SVL labs. According to Echo Bay Minerals, cyanide has not been used in this area and they are not required to monitor for cyanide in LF-I, LF-1, or LF-2. Cyanide detections reported in September and October 1993 may represent false-positives near the detection limit or laboratory blank contamination.

The site LF-I represents sampling from the infiltration and evaporation pond, which receives discharge from the mine adit as well as from the mine shop and wash bay areas.

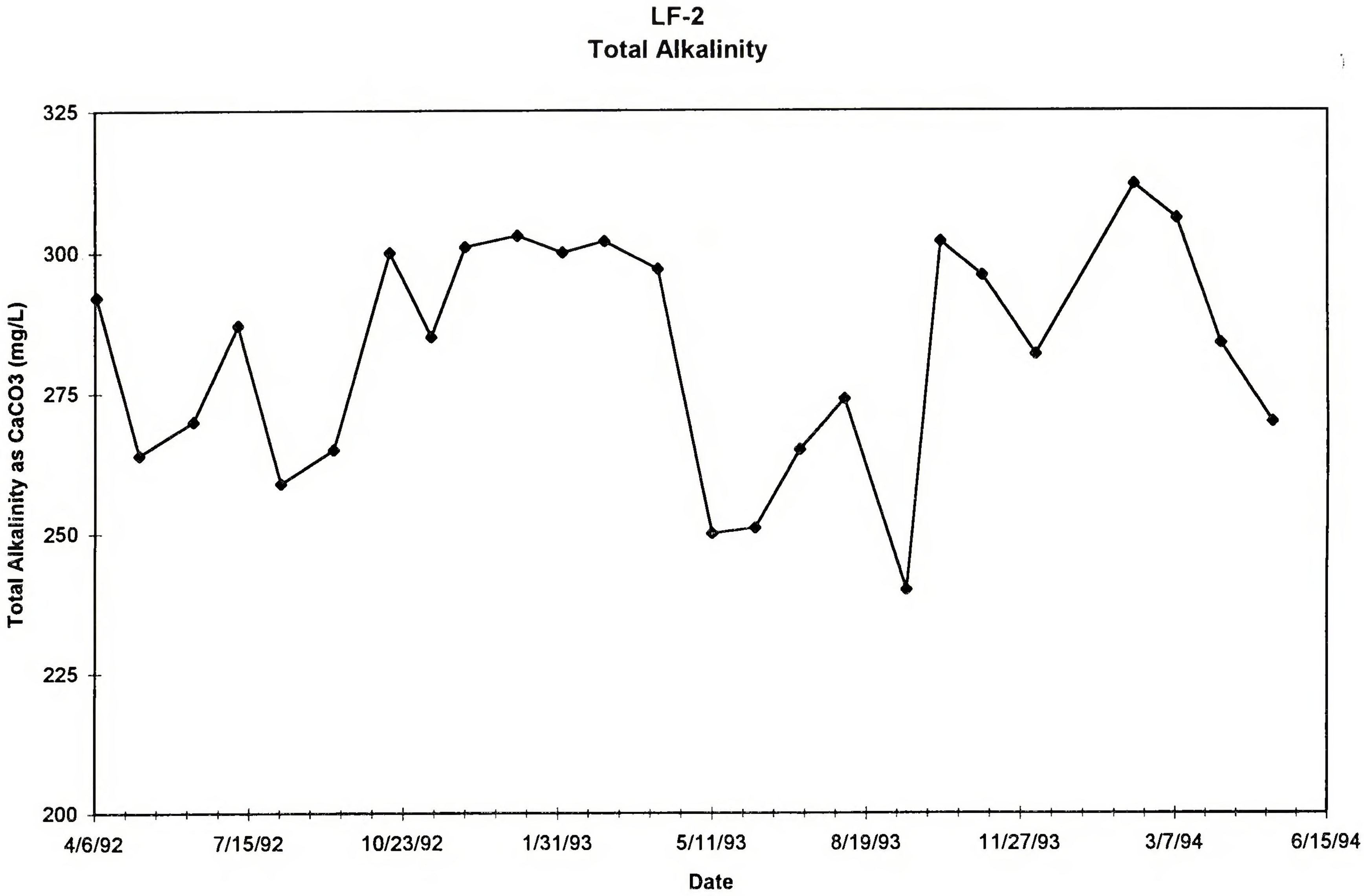

LF-2
Field pH
8.2
8
7.8
7.6
7.4
7.2
7
Field pH
4/6/92
7/15/92
10/23/92
1/31/93
5/11/93
8/19/93
11/27/93
3/7/94
6/15/94
Date

LF2pH.xls

LF-2
Field Specific Conductivity

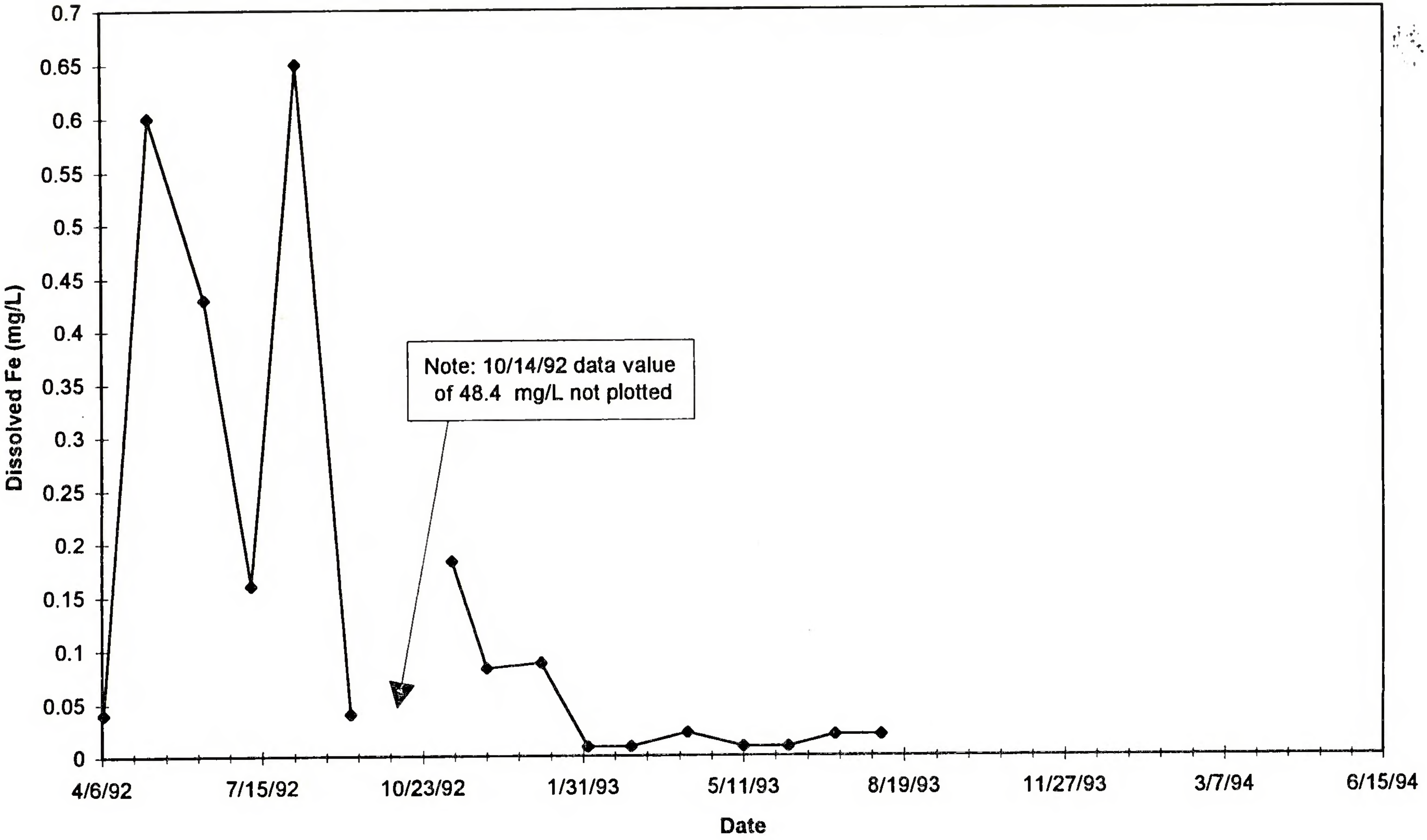


LF2SC.xls



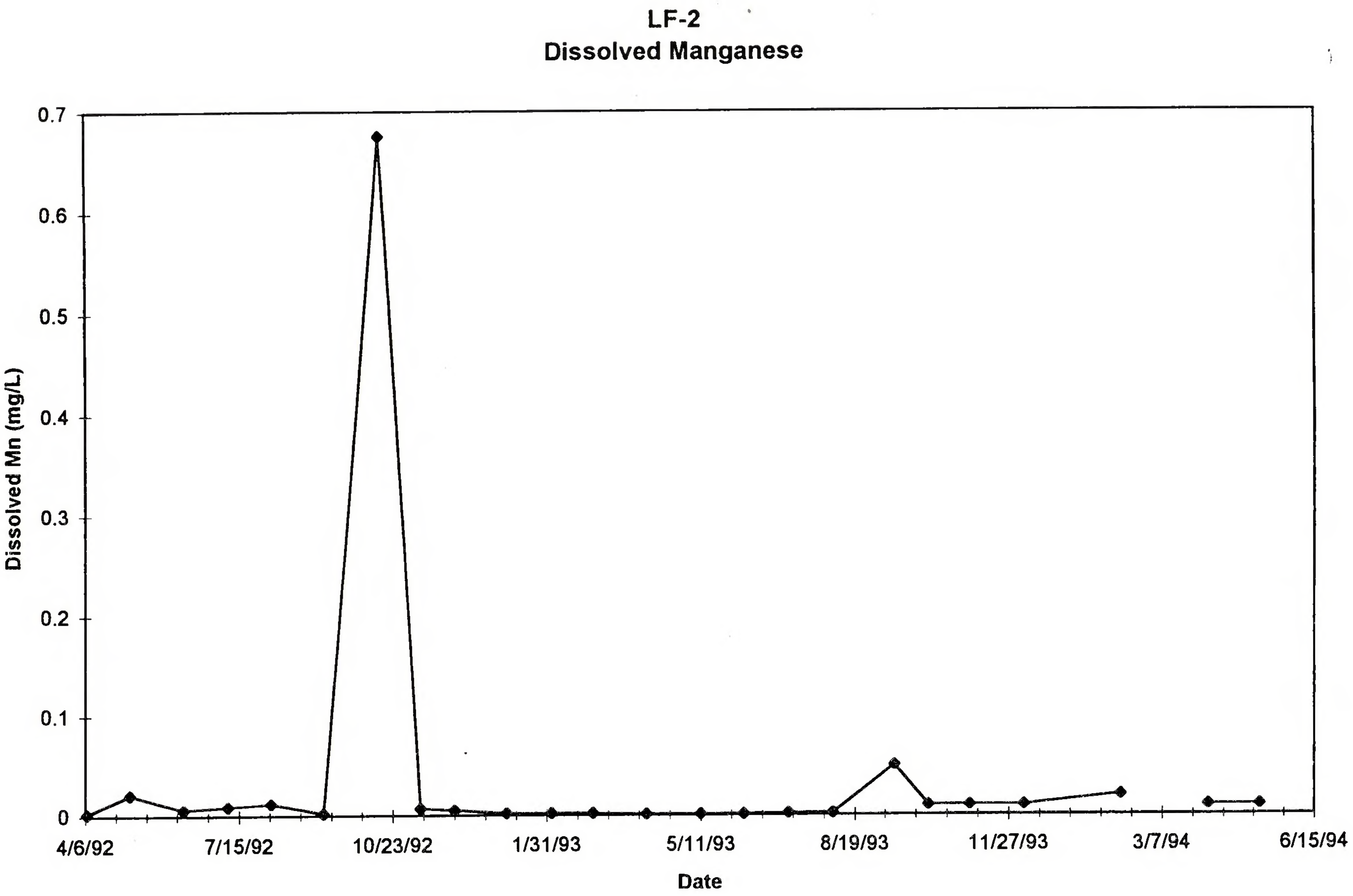
LF2TotAlk.xls

LF-2
Dissolved Iron



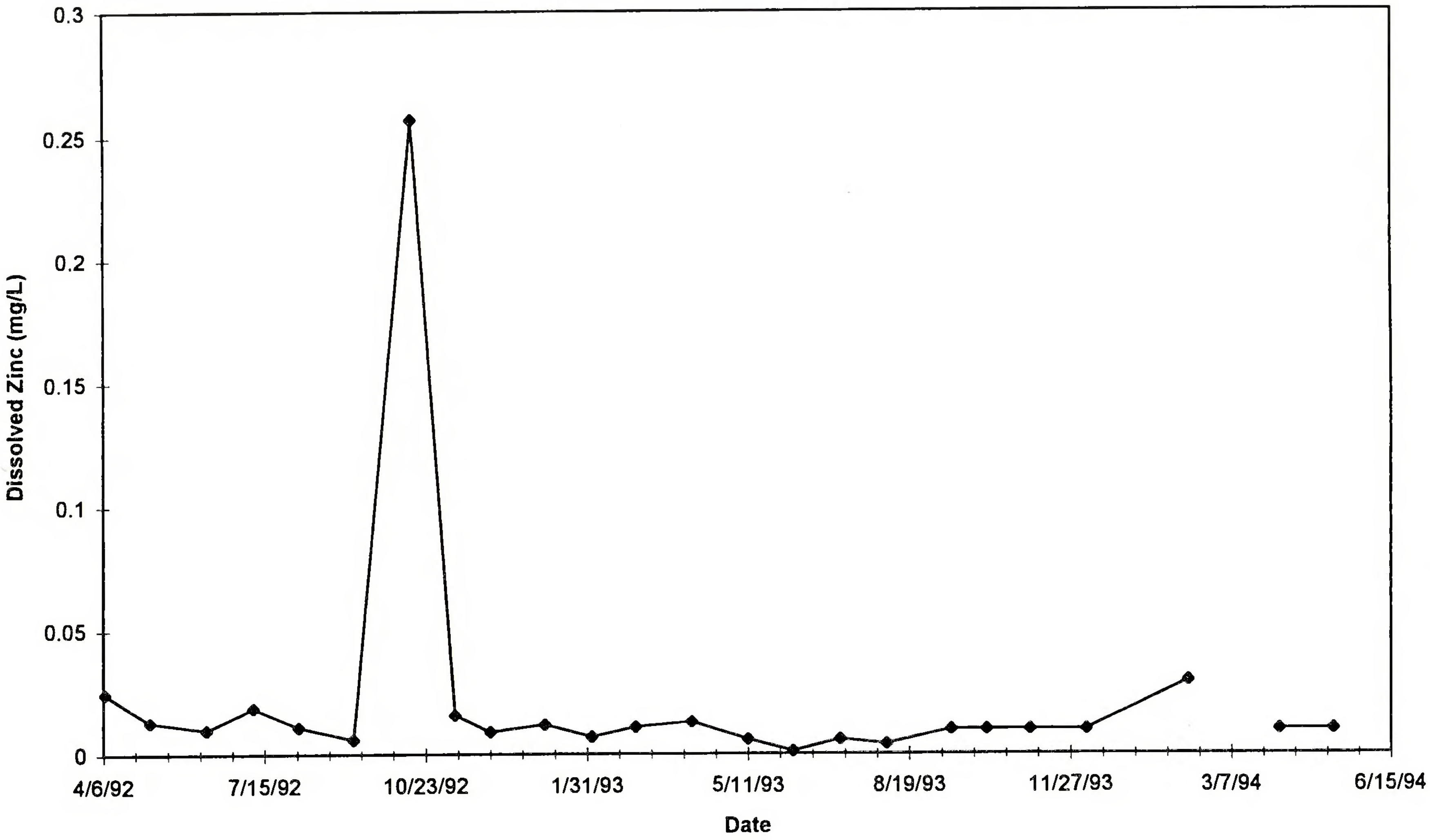
Note: 10/14/92 data value of 48.4 mg/L not plotted

LF2DisFe.xls



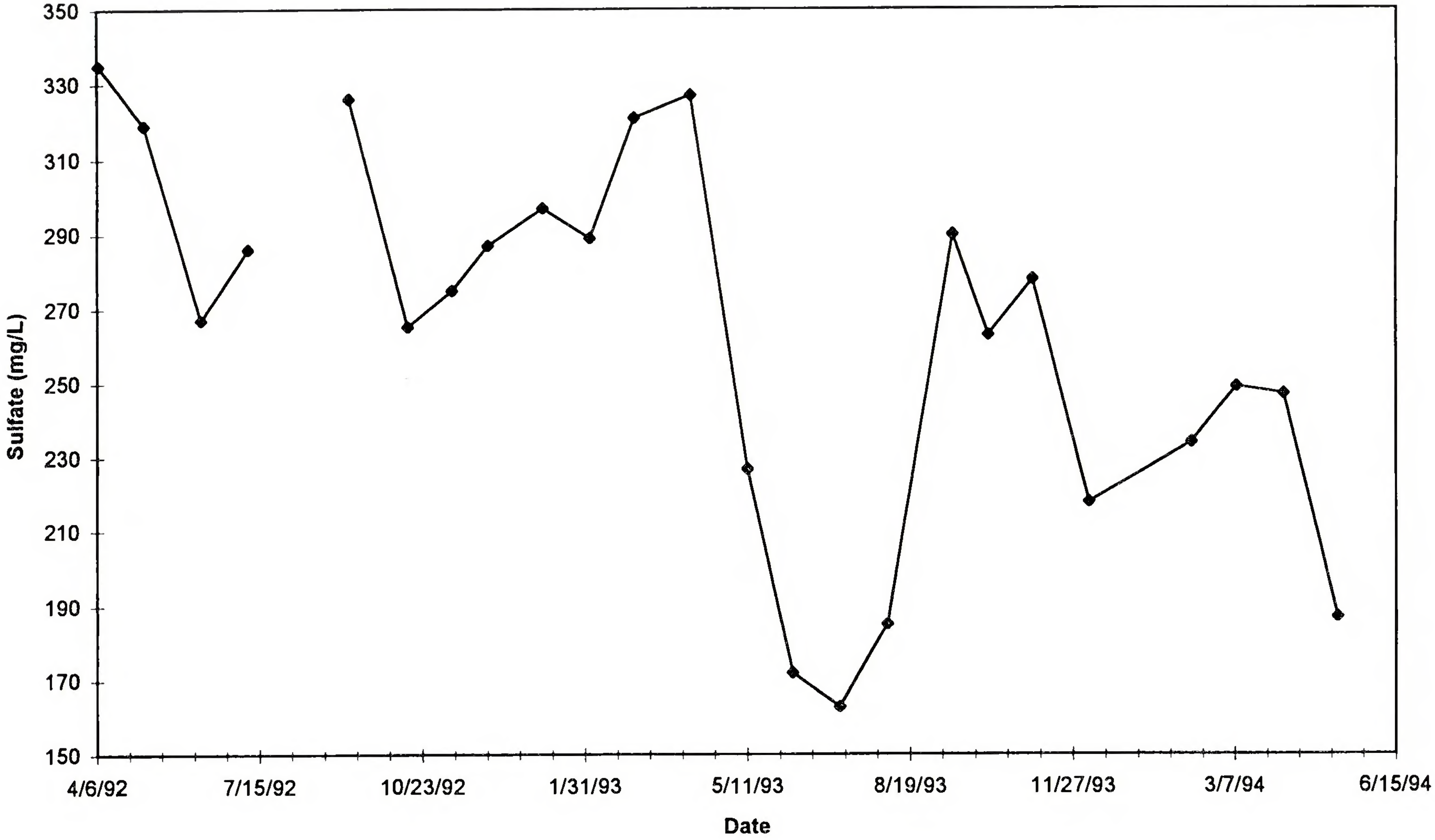
LF2DissMn.xls

LF-2
Dissolved Zinc



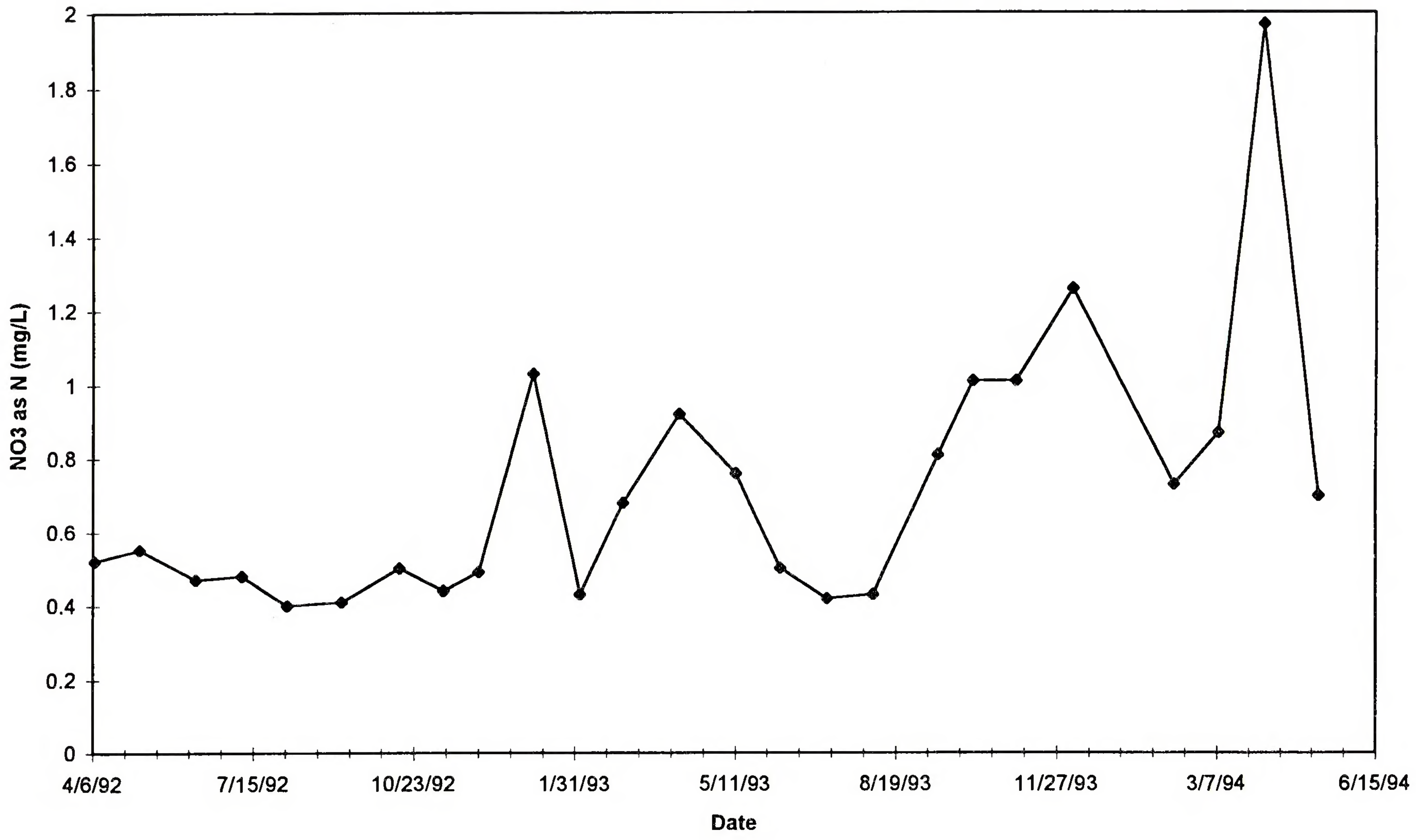
LF2DissZn.xls

LF-2
Sulfate



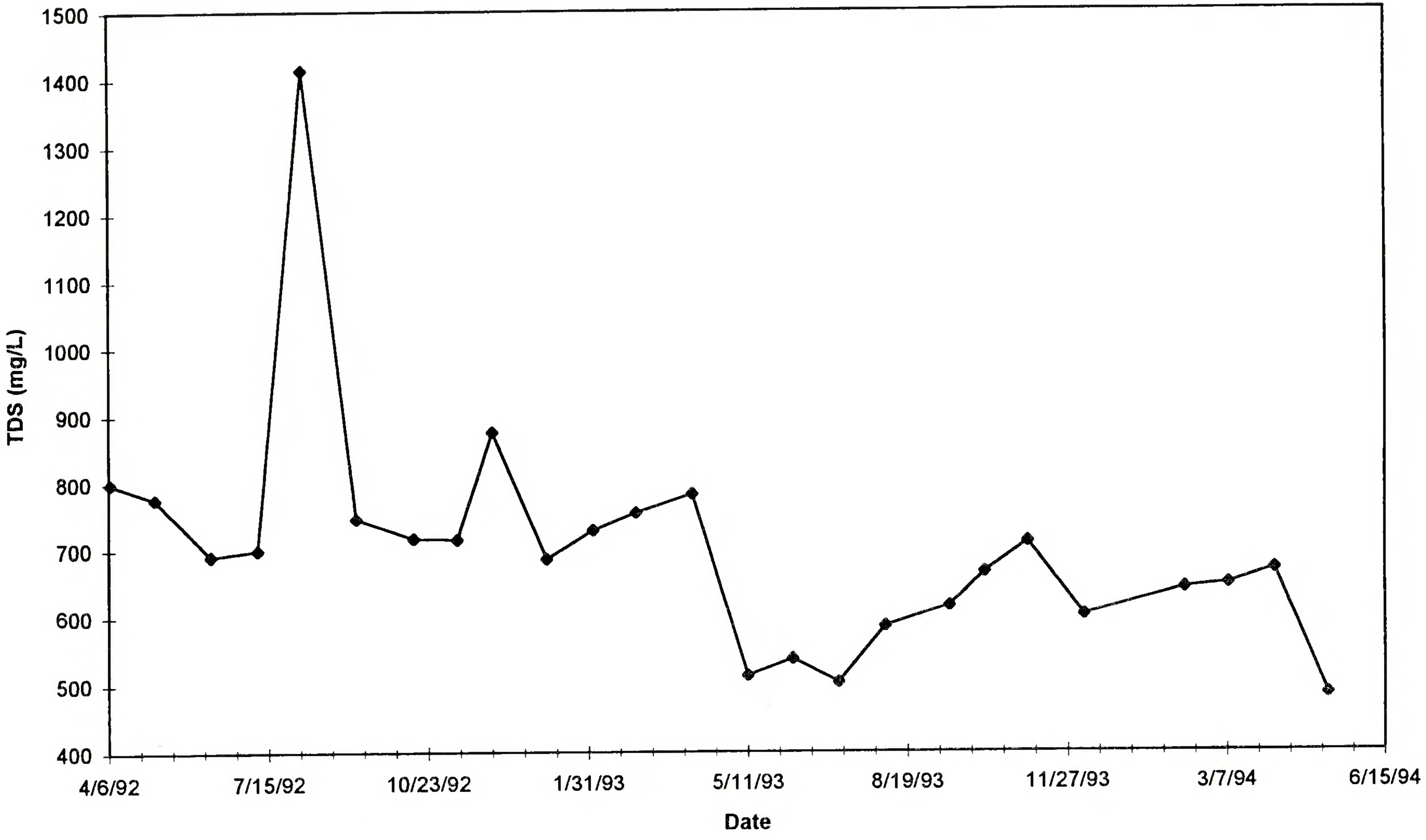
LF2SO4.xls

LF-2
Nitrate as Nitrogen



LF2NO3.xls

LF-2
Total Dissolved Solids



LF2TDS.xls

APPENDIX H
LAMEFOOT GEOCHEMISTRY REPORT
ECHO BAY MINERALS

APPENDIX H
TABLE OF CONTENTS

Section Page

1.0 INTRODUCTION H-1

1.1 OBJECTIVES H-1
1.2 TECHNICAL APPROACH H-1
1.3 HYDROGEOLOGIC SETTING H-2

2.0 INITIAL GEOCHEMICAL CHARACTERIZATION STUDIES H-5

2.1 PHASE I - GEOLOGIC EVALUATION H-5
2.2 PHASE II - IDENTIFICATION AND CHARACTERIZATION OF WASTE AND MINE WALL ROCK LITHOLOGIES H-6

2.2.1 Identification of Rock Lithologies H-6
2.2.2 Waste Rock Volumes and Subsurface Lithologic Distribution H-7

2.3 PHASE III - PRELIMINARY ACID BASE ACCOUNTING ANALYSES H-7

2.3.1 Sample Selection H-8
2.3.2 Acid Base Accounting Procedures H-8
2.3.3 Interpretation of ABA Results H-9
2.3.4 Limitations H-10

2.4 RESULTS H-11
2.5 SUMMARY OF PHASE III PRELIMINARY ACID BASE ACCOUNTING H-12

2.5.1 Waste/Mine Wall Rock Samples H-12
2.5.2 Backfill Samples H-13

3.0 CONTINUING WASTE CHARACTERIZATION PROGRAM H-13

3.1 PHASE IV - SELECTION OF ADDITIONAL MINE WALL ROCK SAMPLES H-14
3.2 PHASE V - SECONDARY ACID BASE ACCOUNTING ANALYSIS H-14
3.3 PHASE VI - IDENTIFICATION OF GEOCHEMICAL DOMAINS H-16
3.4 PHASE VI CONCLUSIONS H-16
3.5 PHASE VII - LITHOLOGIC AND GEOCHEMICAL CORRELATION WITH KEY EXPANSION H-17

3.5.1 Lithologic Correlation H-17
3.5.2 Geochemical Correlation H-19

4.0 ACID DRAINAGE FROM MINE WALLS AND MITIGATIVE APPROACH H-22

4.1 ACID DRAINAGE FROM MINE WALLS H-22
4.2 DEVELOPMENT OF THE MITIGATIVE APPROACH H-24

Section | Page
4.2.1 Prevention of AMD H-24
4.2.2 Abatement of Acid Mine Drainage H-25

5.0 REFERENCES H-26

Following Page

LIST OF TABLES

TABLE 2-1	SUMMARY OF WASTE/MINE WALL ROCK SAMPLES SUBMITTED FOR PHASE III ACID BASE ACCOUNTING ANALYSIS	H-8
TABLE 2-2	GEOLOGIC LOGS FOR BACKFILL EXPLORATION HOLES	H-8
TABLE 2-3	SELECTED EVALUATION CRITERIA FOR THE LAMEFOOT ACID-BASE ACCOUNTING ANALYSES	H-10
TABLE 2-4	RESULTS OF PHASE III ACID-BASE ACCOUNTING ANALYSIS OF WASTE/MINE WALL ROCK SAMPLES	H-11
TABLE 2-5	RESULTS OF PHASE III ACID BASE ACCOUNTING ANALYSIS OF LAMEFOOT BACKFILL SAMPLES	H-11
TABLE 2-6	ABA RESULTS RELATIVE FOR WASTE/MINE WALL ROCK SAMPLES RELATIVE TO THE EVALUATION CRITERIA	H-12
TABLE 2-7	ABA RESULTS FOR THE BACKFILL SAMPLES RELATIVE TO THE EVALUATION CRITERIA	H-13
TABLE 3-1	SUMMARY OF SAMPLES SUBMITTED FOR PHASE V ACID BASE ACCOUNTING ANALYSES	H-14
TABLE 3-2	ACID BASE ACCOUNTING RESULTS FOR PHASE V TESTING	H-15
TABLE 3-3	SUMMARY OF WASTE ROCK LITHOLOGIES PRESENT AT THE KEY EXPANSION AND LAMEFOOT PROJECTS	H-17
TABLE 3-4	SUMMARY OF FACTORS INFLUENCING POTENTIAL GEOCHEMICAL BEHAVIOR OF THE WASTE/MINE WALL ROCK AT BOTH THE KEY EXPANSION AND LAMEFOOT PROJECTS	H-17

Following Page

LIST OF FIGURES

FIGURE 1-1 FLOW CHART ILLUSTRATING THE WASTE CHARACTERIZATION PROGRAM H-1
FIGURE 1-2 CONCEPTUAL MODEL FOR HYDROGEOLOGY AT THE LAMEFOOT PROJECT AREA H-3
FIGURE 1-3 RESULTS OF MARCH 1993 UNDERGROUND HYDROGEOLOGIC SURVEY H-4
FIGURE 2-1 GEOLOGIC CROSS SECTION ILLUSTRATING THE DISTRIBUTION OF WASTE ROCK LITHOLOGIES H-7
FIGURE 2-2 GEOLOGIC CROSS SECTION ILLUSTRATING THE DISTRIBUTION OF WASTE ROCK LITHOLOGIES H-7
FIGURE 2-3 ESTIMATED WASTE ROCK PERCENTAGES H-7
FIGURE 2-4 DISTRIBUTION OF WASTE ROCK LITHOLOGIES EAST WALL OF HW STOPES H-7
FIGURE 2-5 DISTRIBUTION OF WASTE ROCK LITHOLOGIES WEST WALL OF MAIN STOPES H-7
FIGURE 2-6 DISTRIBUTION OF WASTE ROCK LITHOLOGIES WEST WALL OF HW STOPES H-7
FIGURE 2-7 DISTRIBUTION OF WASTE ROCK LITHOLOGIES EAST WALL OF MAIN STOPES H-7
FIGURE 2-8 EAST WALL STOPES EXPOSURE SHOWING LITHOLOGY AND EXPLORATION/DEVELOPMENT WORKINGS H-7
FIGURE 2-9 WASTE AND MINE WALL ROCK GEOCHEMICAL SAMPLING LOCATION MAP H-8

APPENDIX H
LAMEFOOT GEOCHEMISTRY REPORT ECHO BAY MINERALS
AUGUST 1993

1.0 INTRODUCTION

The Lamefoot waste characterization program was developed as an extension of the efforts implemented for Echo Bay's Key Expansion Project. The waste characterization program completed for the Key Project is discussed fully in Appendix E of the Final Environmental Impact Statement, Kettle River Key Project Expansion (Golder 1992). Figure 1-1 illustrates the various procedures which define the waste characterization program for both the Key Expansion and Lamefoot Projects. These programs represent an iterative approach which incorporated sufficient flexibility to allow modifications as data was obtained.

1.1 OBJECTIVES

The principle objectives of the Lamefoot Project waste characterization program are summarized as follows:

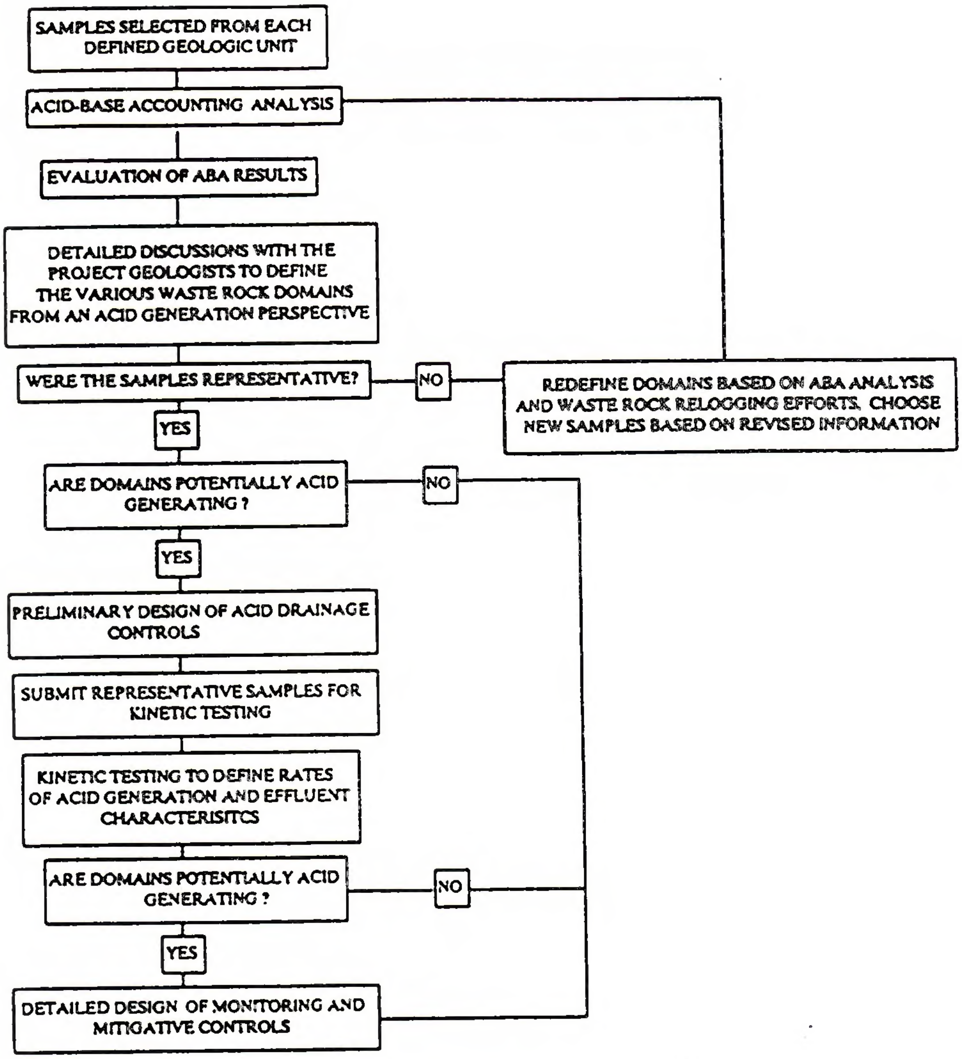
- Predict whether the waste rock generated during mining will potentially produce acid rock drainage (ARD);
- Predict whether the zones of sulfide bearing rock exposed in the underground workings will generate acid mine drainage (AMD);
- Evaluate whether potentially acid generating lithologies and waste rock could introduce contaminants into the environment;
- Optimize the mine closure design; and,
- Optimize the temporary storage and final disposal of the waste removed from the mine.

1.2 TECHNICAL APPROACH

The technical approach developed for the Lamefoot waste characterization program is based on the large geologic and geochemical database which was compiled during waste characterization efforts completed for the Key Expansion Project. The geologic environments of both the Lamefoot and Key Expansion ore deposits are virtually identical therefore, the waste rock domains are also correlative. This geologic similarity allows the full range of geochemical data generated for the Key Expansion to augment the current Lamefoot waste characterization program. One of the initial objectives of the technical approach adopted for the Lamefoot waste rock characterization program involves establishing the degree of variability (both geochemical and geologic) between the Lamefoot and Key Project waste rock domains.

Extensive work was performed as part of the Key Expansion waste characterization program to ensure that the geochemical variability of the various waste rock domains was adequately defined. This process included both acid base accounting and kinetic testing. Preliminary predictions regarding potential acid generation were developed for each of the waste rock domains by evaluating their respective acid base accounts relative to criteria involving the acid neutralizing potential and the acid generation potential (see Appendix E, Golder 1992).

The validity of using acid base accounting (ABA) data to predict the acid generation potential of the various waste rock domains was subsequently confirmed through the kinetic testing completed as part of the Key Expansion waste characterization program. The results of the kinetic testing program illustrated that the


SAMPLES SELECTED FROM EACH DEFINED GEOLOGIC UNIT
ACID-BASE ACCOUNTING ANALYSIS
EVALUATION OF ABA RESULTS
DETAILED DISCUSSIONS WITH THE PROJECT GEOLOGISTS TO DEFINE THE VARIOUS WASTE ROCK DOMAINS FROM AN ACID GENERATION PERSPECTIVE
WERE THE SAMPLES REPRESENTATIVE?
NO
REDEFINE DOMAINS BASED ON ABA ANALYSIS AND WASTE ROCK RELOGGING EFFORTS. CHOOSE NEW SAMPLES BASED ON REVISED INFORMATION
YES
ARE DOMAINS POTENTIALLY ACID GENERATING ?
NO
YES
PRELIMINARY DESIGN OF ACID DRAINAGE CONTROLS
SUBMIT REPRESENTATIVE SAMPLES FOR KINETIC TESTING
KINETIC TESTING TO DEFINE RATES OF ACID GENERATION AND EFFLUENT CHARACTERISITCS
ARE DOMAINS POTENTIALLY ACID GENERATING ?
NO
YES
DETAILED DESIGN OF MONITORING AND MITIGATIVE CONTROLS

Golder Associates
Denver, Colorado

TITLE

FLOW CHART ILLUSTRATING THE WASTE CHARACTERIZATION PROGRAM

CLIENT/PROJECT
ECHO BAY MINERALS Co.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

DRAWN	KAW	DATE	AUG 1993	JOB NO.	933-2614
CHECKED	JC	SCALE		DWG NO./REV. NO.	
REVIEWED	JC	FILE NO.		FIGURE NO.	1-1

criteria established to define acid generation potential, based on ABA data for the various waste rock domains, were representative of actual geochemical behavior and appropriately conservative.

The geologic/lithologic similarity between the two sites, and the usefulness of ABA data to properly predict anticipated geochemical behavior of the various waste rock domains provide the focus for the Lamefoot waste characterization program. Much of the focus of the Lamefoot program is centered on establishing both the geologic/geochemical similarities and variabilities between the waste rock domains present at both sites in order to facilitate utilization of existing data and interpretations to direct the Lamefoot Project geochemical program and waste disposal design.

The technical approach developed for the Lamefoot waste characterization program can be summarized as follows:

- Confirm the lithologic/geologic similarity between the waste rock domains present at both the Lamefoot and Key Expansion projects;
- Establish the geochemical variability, if any, between the waste rock domains common to both the Lamefoot and Key Expansion projects; and,
- Identify the waste rock domain(s) present at the Lamefoot project with the highest potential for acid generation and bias future sampling and geochemical testing to establish the variability in acid generation potential within these domains.

This technical approach has been implemented through a seven phase waste characterization program consisting of the following elements:

- Phase I Geologic Evaluation.
- Phase II Identification and Characterization of Mine and Waste Rock Lithologies.
- Phase III Preliminary Acid Base Accounting Analysis.
- Phase IV Selection of Additional Mine Rock Samples for Secondary Geochemical Characterization.
- Phase V Secondary Acid Base Accounting Analysis.
- Phase VI Definition of Geochemical Mine and Waste Rock Domains.
- Phase VII Correlation with Key Expansion Waste Characterization Program.

The first three phases represent the initial step in an iterative waste characterization program which focused on providing a lithologic and geochemical foundation. Phases IV through VII represent the secondary elements of the program and focus on defining more clearly the potential geochemical variability and behavior of the waste rock domains.

1.3 HYDROGEOLOGIC SETTING

The Lamefoot Project site has two general types of groundwater occurrence. One is the uppermost water table which occurs within the unconsolidated alluvial and glacial deposits at topographic lows and along the flatland at the mountain front. At higher elevations and on mountain side slopes, this uppermost groundwater occurrence is irregular and controlled by the residual soils/bedrock contact within both the soils and the more fractured weathered bedrock surface. The groundwater at higher elevations flows downgradient and is controlled primarily by the topography of the bedrock surface. Most precipitation recharges the shallow alluvial/glacial unit and discharges from the site as near-surface alluvial groundwater. To a lesser extent, a portion of the shallow groundwater percolates into fractures to recharge the underlying unweathered bedrock.

The second type of groundwater at the Lamefoot Project site occurs within the deeper fractured bedrock (e.g., at the depths of the current Lamefoot mine). Bedrock groundwater occurs locally within a small number of water bearing fractures. Although only small amounts of groundwater have been encountered in the bedrock at the site, there is the potential for higher flows where large, interconnected, open fractures occur. No such large fractures have been evidenced or suggested in the workings to date or in close spaced exploration drilling. An exception is the relatively high flow rate (50 gpm) produced from the water supply well at the facilities area that intercepts a zone of highly fractured shale bedrock at a depth of 265 feet below ground surface. The water supply well was located and drilled to encounter the geologic structural control for the Wolfe Camp valley and the range front. This water bearing fracture zone is interpreted to represent a northerly trending structure believed to be unrelated to the mineralized fracture zones. The Lamefoot underground workings to date have not encountered related water bearing fracture zones.

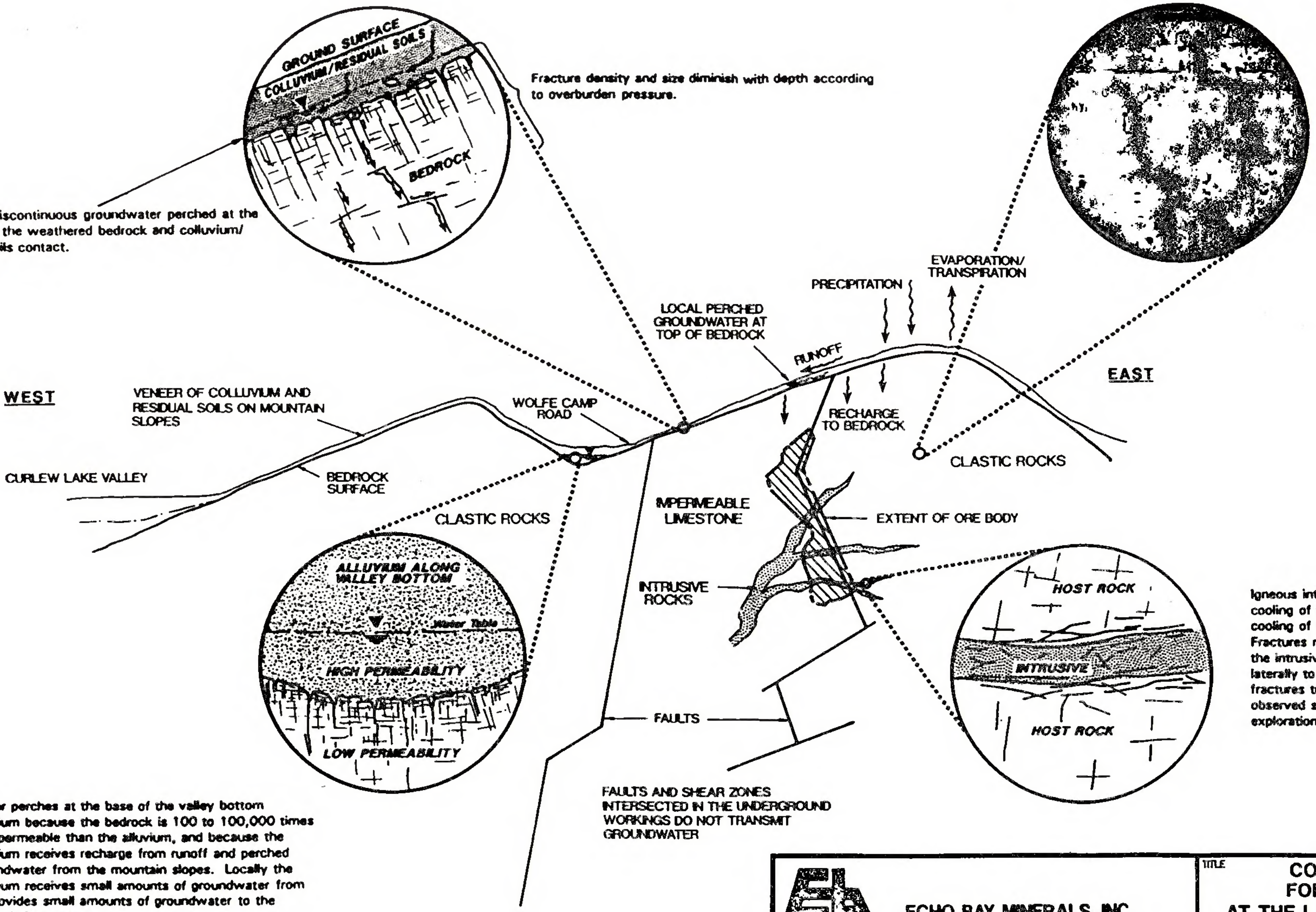
Site experience shows that some drill holes may be dry, while others are wet where water bearing fractures are intercepted. Similarly, in the underground exploration and development workings, isolated portions of the mine walls are wet or dripping while the majority of the walls are dry. Figure 1-2 presents a conceptual diagram of principal components of the hydrogeology of the Lamefoot Project site.

Bedrock geology controls the occurrence of groundwater within the Lamefoot mine. Because fractures are the sole mechanism for flow, bedrock fracturing as related to rock type, mineralization, structure (faults), and truncation by intrusives strongly determines the location and magnitude of groundwater flow.

As observed during both the surface drilling and underground exploration programs, bedrock comprised of unaltered limestone generally yields no water. Even where fractured, the unaltered limestone contains little or no water, suggesting that fractures in this unit are not sufficiently extensive and/or interconnected to carry groundwater. In contrast, the massively mineralized limestone (i.e., low grade ore) which is well fractured and jointed carries small quantities of groundwater. Fractures associated with the rhyodacite intrusive rocks tend to yield the more significant flows observed underground and during surface drilling. These intrusive-related water bearing fractures have been noted in both the rhyodacite itself and in the host rock immediately surrounding the intrusion. Though the intrusion related fractures produce the most significant water encountered in the bedrock, quantities are nonetheless small. Of the several visibly flowing intrusive-related fractures observed underground, all produced water at less than 0.08 gpm from a single fracture or fracture set.

The elevation of the phreatic surface within the bedrock varies according to the degree of interconnectedness, orientation, and spacing of the fractures that makeup the fracture system. The degree of connection a particular fracture or fracture set has with the surface is also an important factor which controls the phreatic surface. Because of variability in fracture characteristics the phreatic surface can vary dramatically between the localized fracture sets (see Figure 1-1).

The fact that no springs occur on the mountain slopes and that only small quantities of inflow are observed in the mine suggest small amounts of recharge to the bedrock. At no time during the exploration mining program has groundwater discharged into the workings at high flow rates or pressures (i.e., pressurized inflow on a face immediately following blasting).



Fracture density and size diminish with depth according to overburden pressure.

Thin and discontinuous groundwater perched at the contact of the weathered bedrock and colluvium/ residual soils contact.

The elevation of the phreatic surface in the bedrock is highly dependent on the spacing, orientation, size, and interconnectedness of the fracture network. Therefore, the phreatic surface varies from location to location. The degree of connection of the fracture network to the surface for recharge by precipitation is a critical factor.

Igneous intrusive rock fractures are created from the cooling of the intrusion and from the heating and cooling of the immediately surrounding host rock. Fractures may be extensive and interconnected along the intrusive/host rock contact, but are confined laterally to the immediate area of the intrusive. These fractures transmit the majority of groundwater observed seeping into the underground exploration/development workings.

FAULTS AND SHEAR ZONES INTERSECTED IN THE UNDERGROUND WORKINGS DO NOT TRANSMIT GROUNDWATER

Water perches at the base of the valley bottom alluvium because the bedrock is 100 to 100,000 times less permeable than the alluvium, and because the alluvium receives recharge from runoff and perched groundwater from the mountain slopes. Locally the alluvium receives small amounts of groundwater from or provides small amounts of groundwater to the bedrock fracture system.

ECHO BAY MINERALS, INC.

TITLE
CONCEPTUAL MODEL FOR HYDROGEOLOGY AT THE LAMEFOOT PROJECT AREA

CLIENT/PROJECT
ECHO BAY MINERALS, INC.
LAMEFOOT PROJECT

DRAWN LC	DATE JUNE 1993	JOB NO. 933-2614
CHECKED	SCALE NOT TO SCALE	DWG. NO./REV. NO.
REVIEWED	FILE NO.	FIGURE NO. 1-2

Inflow to Existing Workings

Groundwater inflow rates to the underground exploration/development workings are currently being monitored on a monthly basis. In addition, an evaluation of the inflow as it relates to rock lithologies and structure was performed during the week of March 15, 1993. At that time, wet areas, drips, and flowing water were mapped, flow rates estimated, samples collected for laboratory analysis, and field tests performed for pH, electrical conductance and temperature. Figure 1-3 presents the results of that evaluation. Water inflow rates were estimated and field water quality tests were taken from four observed sources of groundwater inflow:

- Inflow from all 10 exploration boreholes drilled from the surface which were intersected by the workings;
- Occasional inflow from horizontal exploration boreholes drilled into the drift walls and vertical boreholes drilled into the drift roof into which were installed rock bolts for securing roof straps;
- Rare to occasional inflow from fractures associated with dikes and sills cutting the rock fabric; and,
- Occasional to common inflow from highly altered and mineralized ore-bearing fractured/jointed micritic limestone.

Mine inflow was limited to discrete or more highly fractured areas, the largest of which occurred over an area estimated at 100 square feet. Areas of unaltered limestone were commonly less fractured, and produced little water. This suggests that the fractures in the unaltered limestone are not extensively interconnected and are not subject to surficial recharge. At the time of the underground evaluation, no extensive areas of clastie rock were exposed. On those limited clastic exposures no wet areas were observed. A discussion of the waters encountered by the type of source is presented below.

Intersected Boreholes

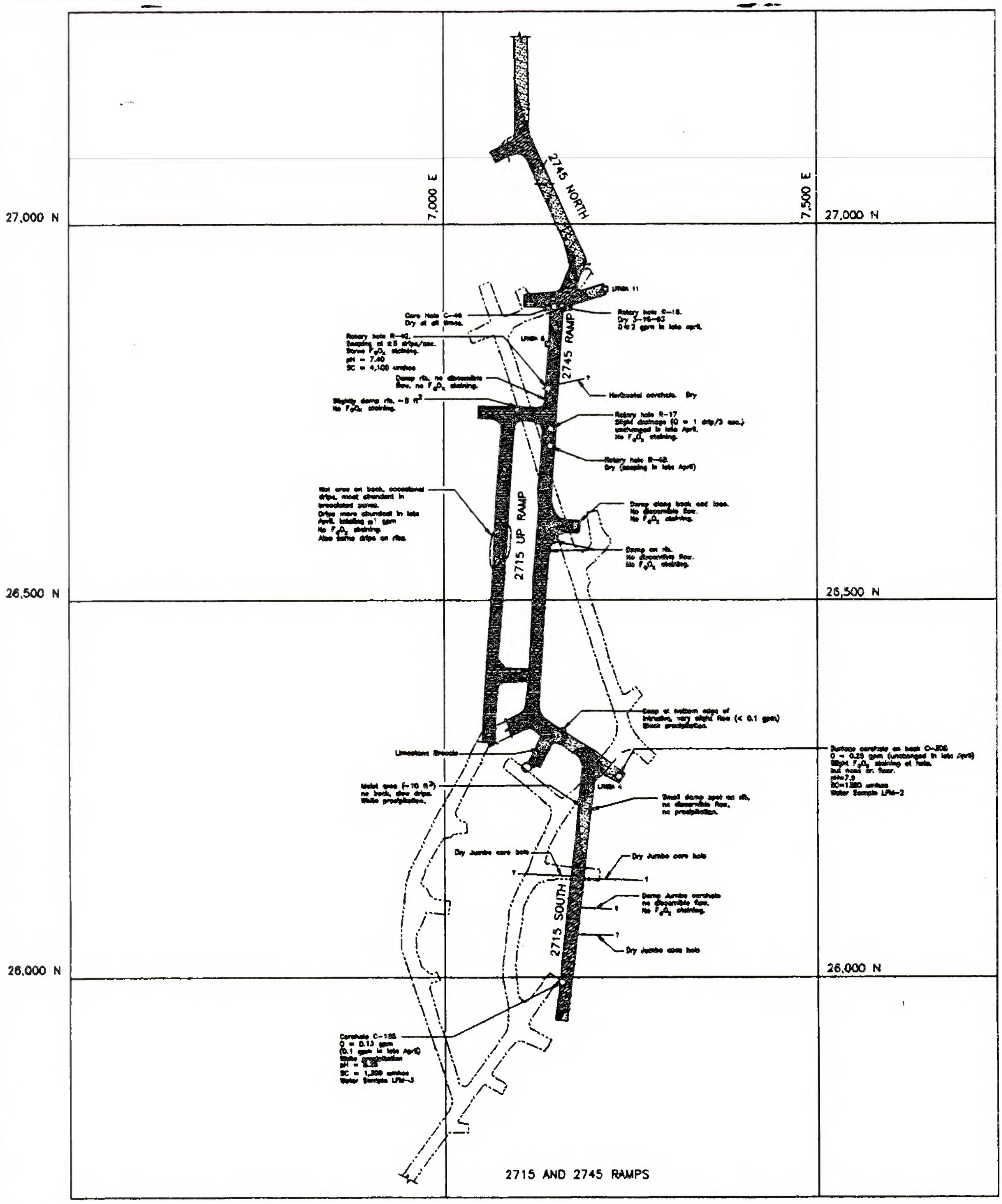
The largest inflows observed in the workings were noted where surface-drilled boreholes were intersected by the workings. Of the 10 boreholes encountered five had measurable discharge, four were dry, and one had a discharge rate too low to measure. Maximum flow from a single borehole was estimated at 2 gpm, and total inflow from the five flowing boreholes was an estimated 2.38 gpm during the March 1993 evaluation. This represents the highest inflow observed in the workings. Because the boreholes are currently open to a depth near the surface, the exact source of water cannot be identified with confidence. The source could be from direct precipitation/snow melt inflowing from the surface, discharge of shallow perched saturation in the residual soils and weathered upper bedrock, or from deeper fractures.

Horizontal Boreholes Into Drift Walls

Horizontal exploration boreholes drilled into the mine walls were generally dry during the March 1993 underground evaluation. Where wet, seepage rates were very low with little or no visible flow.

Rockbolt Boreholes

The vast majority of rockbolt boreholes were dry at the time of the underground evaluation. Those that showed evidence of moisture were generally located in or adjacent to the intrusive rocks.



LEGEND:

- ▨ CLASTICS (UNDIFFERNTIATED)
- ■ LIMESTONE
- ■ INTRUSIVES
- ■ ALTERED LIMESTONE
- ■ LIMESTONE BRECCIA
- ---• HORIZONTAL COREHOLE INTO RIB. '?' INDICATES UNKNOWN DEPTH
- ▪ ACID BASE ACCOUNTING SAMPLE LOCATION
- ◇ SURFACE BOREHOLE INTERCEPT
- ⁄ FAULTS/MAJOR JOINTS
- ⁄ INTRUSIVE DIKE

NOTES:

1. SHOWN IS THE EXTENT OF GEOLOGIC MAPING AS OF MARCH 1993 EXCEPT WERE NOTED. THE UNDERGROUND WATER INFLOW CONDITIONS OBSERVED MARCH 16, 1993.
2. WATER QUALITY MEASURED USING FIELD pH AND SPECIFIC CONDUCTANCE METER.
3. ACID BASE ACCOUNTING (ABA) SAMPLES LFABA-7, LFABA-8, LFABA-9 AND LFABA-10 ARE LOCATED IN WORKINGS DRIVEN AFTER MARCH 1993 AND ARE NOT SHOWN.
4. WATER SAMPLES LFM-4 WAS COLLECTED IN THE ACCESS TUNNEL. LFM-5 WAS COLLECTED IN THE MINE SUMP WATER NEAR THE PORTAL.

Eb

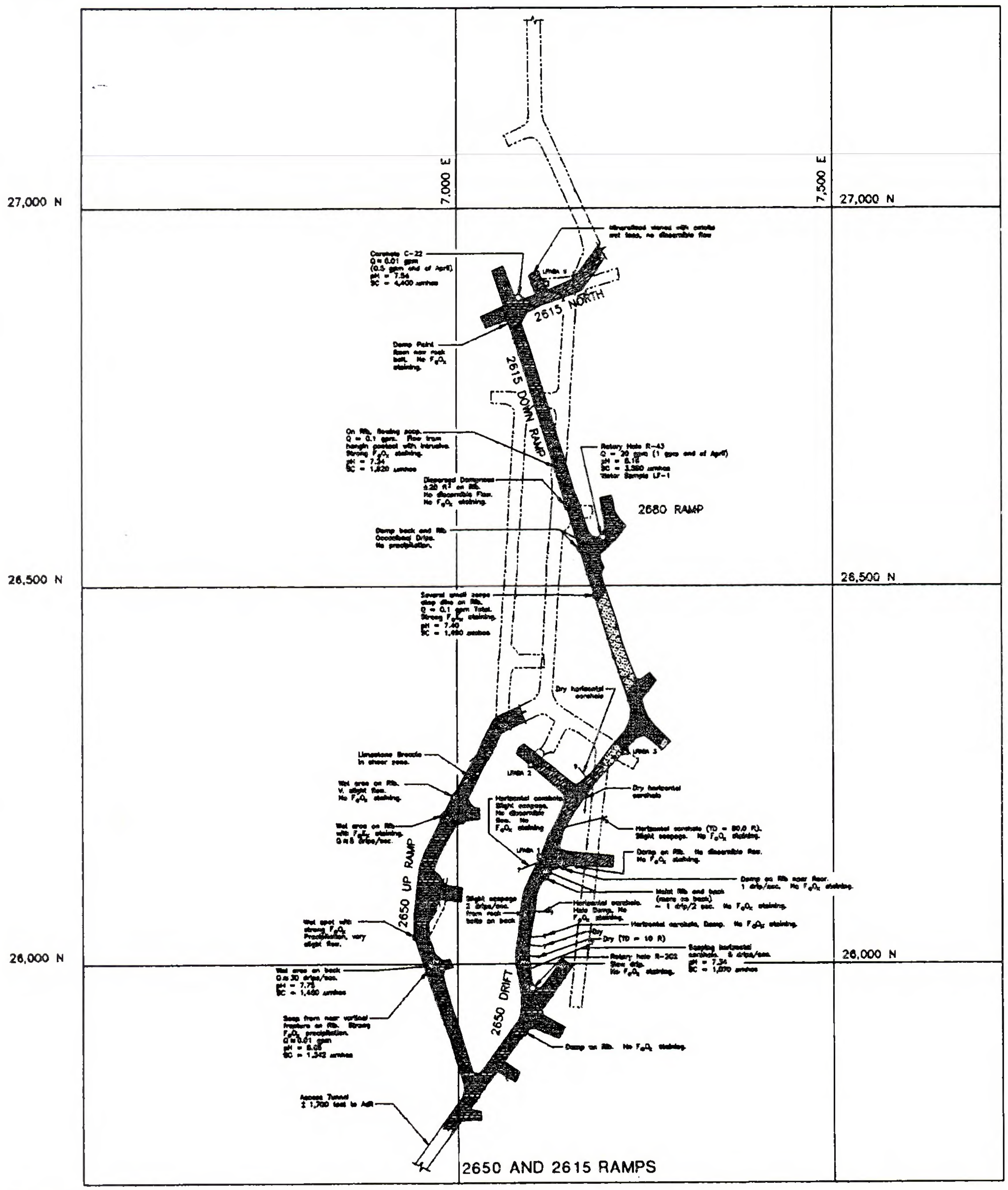
ECHO BAY LAMEFOOT PROJECT

Source: Adapted from Echo Bay 1993a

Echo Bay Minerals Co.
Lamefoot Mine
Kettle River Operations
Republic, Washington

RESULTS OF MARCH 1993
UNDERGROUND
HYDROGEOLOGIC SURVEY

DATE JUNE 1993 SCALE FIG NO. 1-3A



LEGEND:

- ▨ CLASTICS (UNDIFFERNTIATED)
- ■ LIMESTONE
- ■ INTRUSIVES
- ■ ALTERED LIMESTONE
- ■ LIMESTONE BRECCIA
- —· HORIZONTAL COREHOLE INTO RIB. '?' INDICATES UNKNOWN DEPTH
- ● ACID BASE ACCOUNTING SAMPLE LOCATION
- ◇ SURFACE BOREHOLE INTERCEPT
- ⁄ FAULTS/MAJOR JOINTS
- ⁄ INTRUSIVE DIKE

NOTES:

1. SHOWN IS THE EXTENT OF GEOLOGIC MAPING AS OF MARCH 1993 EXCEPT WERE NOTED. THE UNDERGROUND WATER INFLOW CONDITIONS OBSERVED MARCH 16, 1993.
2. WATER QUALITY MEASURED USING FIELD pH AND SPECIFIC CONDUCTANCE METER.
3. ACID BASE ACCOUNTING (ABA) SAMPLES LFABA-7, LFABA-8, LFABA-9 AND LFABA-10 ARE LOCATED IN WORKINGS DRIVEN AFTER MARCH 1993 AND ARE NOT SHOWN.
4. WATER SAMPLES LFM-4 WAS COLLECTED IN THE ACCESS TUNNEL. LFM-5 WAS COLLECTED IN THE MINE SUMP WATER NEAR THE PORTAL.

ECHO BAY LAMEFOOT PROJECT

Source: Adopted from Echo Bay 1993a

Echo Bay Minerals Co.
Lamefoot Mine
Kettle River Operations
Republic, Washington

RESULTS OF MARCH 1993
UNDERGROUND
HYDROGEOLOGIC SURVEY

DATE JUNE 1993 | SCALE | FIG. NO. 1-3B

SH2

Low Grade Ore (Mineralized Limestone)

Wet conditions observed in the low grade ore zones were generally associated with dispersed, larger areas (25 to 100 square feet) of moisture which occurred on walls or roofs. However, no concentrated flow (drips, rivulets, etc.) was observed to occur within these areas. The more fractured, jointed and broken-up nature of the mineralized areas in the limestone may be the cause of the dispersed low inflow rates.

Outflow from Existing Workings

At the time of the underground evaluation the total inflow to the mine was estimated to be on the order of 25 to 35 gpm. This estimate includes all sources including the surface exploration boreholes and return drill water and was based on the inflow to the underground sump located approximately 280 feet inside the portal. The entire volume of underground drainage enters the sump and is ultimately reused as drill water for the ongoing exploration/development program. The water flow to the sump via the drainage ditches, which are excavated along the side of all drifts and ramps, includes both mine inflow and water pumped underground from the water supply well located in the facilities area. Based on the varying amounts of drilling water used at any time, and on the attenuation of flow along the more than 3,000 feet of drainage ditch (cumulative ditch length of more than 6,500 feet), a precise assessment of natural mine inflow is tenuous. However, based on seep flow data obtained and observations made during the underground evaluation of mid March, 1993, natural fracture inflow (i.e., not from surface boreholes or drill water) was estimated at 10 to 15 gpm.

Flow has decreased during the summer months and measured less than 10 gpm at its lowest point. Echo Bay geologists continue to map and monitor groundwater occurrences within the workings during the on-going exploration program.

2.0 INITIAL GEOCHEMICAL CHARACTERIZATION STUDIES

During the early planning stages of the Lamefoot Project, initial studies were completed to ensure that the on-going program would have the focus necessary to achieve the appropriate level of geochemical characterization. The initial stage of the Lamefoot geochemical characterization program is defined by Phases I through III:

- Phase I Geologic Evaluation;
- Phase II Identification and Characterization of Waste Rock Lithologies; and,
- Phase III Preliminary Acid Base Accounting Analysis.

The sections which follow provide a summary of the technical approach and results of the initial geochemical characterization efforts.

2.1 PHASE I - GEOLOGIC EVALUATION

As part of the internal pre-feasibility study completed by Echo Bay, a geologic evaluation of the waste rock, mine wall rock and backfill was initiated. Drill core collected in 1990 was logged by Echo Bay geologists to record geological properties including rock type, mineralogy, sulfide mineral abundance and distribution, the presence of fractures, and other characteristics which may influence geochemical behavior. The features noted during the core logging are summarized below:

- Lithology
- Alteration
- Mineralization
  - % Sulfides

  - Type
  - Style
  - Intensity
- Structure
  - Bedding
  - Faults/Shears
  - Fractures/Joints
  - Veins

  - Occurrence
  - Size
    - Habit
  -% Carbonates
    - Occurrence
  - Abundance
    - Reactivity

The geologic information was used to identify the various lithologies, and qualitatively identify the physical and geochemical properties of the waste rock and mine wall exposures which may influence effective surface and subsurface waste management. A major goal of the geologic evaluation was to establish preliminary lithologic classification of the rock types present at the Lamefoot Project.

Initial identification of waste/mine wall rock domains is generally based on lithologic subdivision, however the preliminary classification developed for Lamefoot was designed to identify the spatial variation in mineralization or alteration within each lithologic unit. In this manner, the range of potential variability in geochemical behavior will be evaluated. Thus, the preliminary geologic evaluation will allow the on-going geochemical testing program to focus on those domains which exhibit the greatest potential for both variability and acid generation.

In addition, underground mapping was initiated in 1991 to gather similar geologic information needed to construct geologic cross sections and longitudinal stope sections. The sections were developed to define the relative distribution and areas of exposure for the lithologies within the proposed stopes.

The following studies were also initiated for internal planning purposes;

- **Ore and Metallurgical Studies** - evaluation of process reagents and products, precious element mineralogy and distribution, deleterious trace element mineralogy, and metallurgical factors affecting concentrate grade, detailed evaluation of sulfide mineralogy, mineral composition, and ore textures.

- **Petrographic studies** - of the core samples were also performed to assist in mineral identification, and the evaluation of ore and mineral alterations.

The metallurgical and mineralogical data were used to further characterize the mode of occurrence, habit, and other factors which potentially influence the geochemical behavior of the sulfide minerals. The data collected during this phase was used to provide the appropriate focus to the future phases of the Lamefoot waste characterization program in order to achieve effective implementation.

2.2 PHASE II - IDENTIFICATION AND CHARACTERIZATION OF WASTE AND MINE WALL ROCK LITHOLOGIES

2.2.1 Identification of Rock Lithologies

Based on the preliminary geologic investigations performed by Echo Bay, four general rock types were identified at the Lamefoot Project: limestone, low grade ore, intrusive, and clastic rocks. Lithologic descriptions for each of the rock types are presented below:

> **Limestone:** Light to dark gray, massive with no apparent bedding, sandy to micritic in texture, locally sparitic, weak to moderate argillization, sulfides consist mainly of pyrite, sulfide occurrence

is limited to veins and veinlets. Sulfide habit is generally amorphous to euhedral, and visual quantities are estimated to range from trace to five percent.

Low Grade Ore: Dark greenish gray and brownish gray mottled altered limestone, sandy to micritic in texture, occasionally dolomitic, mineralized zones contain quartz veins and/or pervasive silicification with significant sulfides, magnetite, and/or hematite. Sulfide minerals consist primarily of pyrite which ranges in occurrence from massive to disseminated. Cumulative percentages of sulfide, magnetite, and/or hematite generally exceed 5 percent and in some areas can exceed 50 percent.

Clastic: Siliciclastic rock ranging from siltstone to sharpstone conglomerate, containing both greenstone and localized carbonate intervals. Sulfides consist primarily of pyrite and range from less than one percent to 10 percent depending on proximity to mineralization. Pyrite occurs mainly in veinlets and fractures with both calcite and quartz.

Intrusive: Porphyritic rhyodacite with biotite or hornblende. Alteration ranges from propylitic to argillitic with calcite and chlorite as common alteration products. Sulfide percentages range from zero to 0.1 percent and typically are disseminated throughout the groundmass.

Mine Backfill: Primarily composed of clastic and intrusive rock. Clastic rocks dominate and are siliciclastic siltstones, with variable amounts of sulfide and carbonate mineralization. Portions of the clastic rocks exhibit significant degrees of oxidation which generally decreases with depth. Visual sulfide percentages range from trace to 2 percent. Sulfide occurrence is primarily as fine grained disseminations within the matrix and quartz veins. The intrusive rocks are fine grained, with phenocrysts of biotite and feldspar, trace calcite and pyrite occur within veinlets and the groundmass.

The basic geologic environment of the Lamefoot deposit consists of a mineralized zone along the steeply dipping mineralized and healed Anfo fault. The fault is bounded on the east by clastic rocks and both unaltered and low grade ore (altered limestone) on the west. Economic mineralization extends into both the clastic and altered limestone rocks to varying degrees. Figures 2-1 and 2-2 illustrate typical cross sections near the north and south limits of the proposed mining area, respectively.

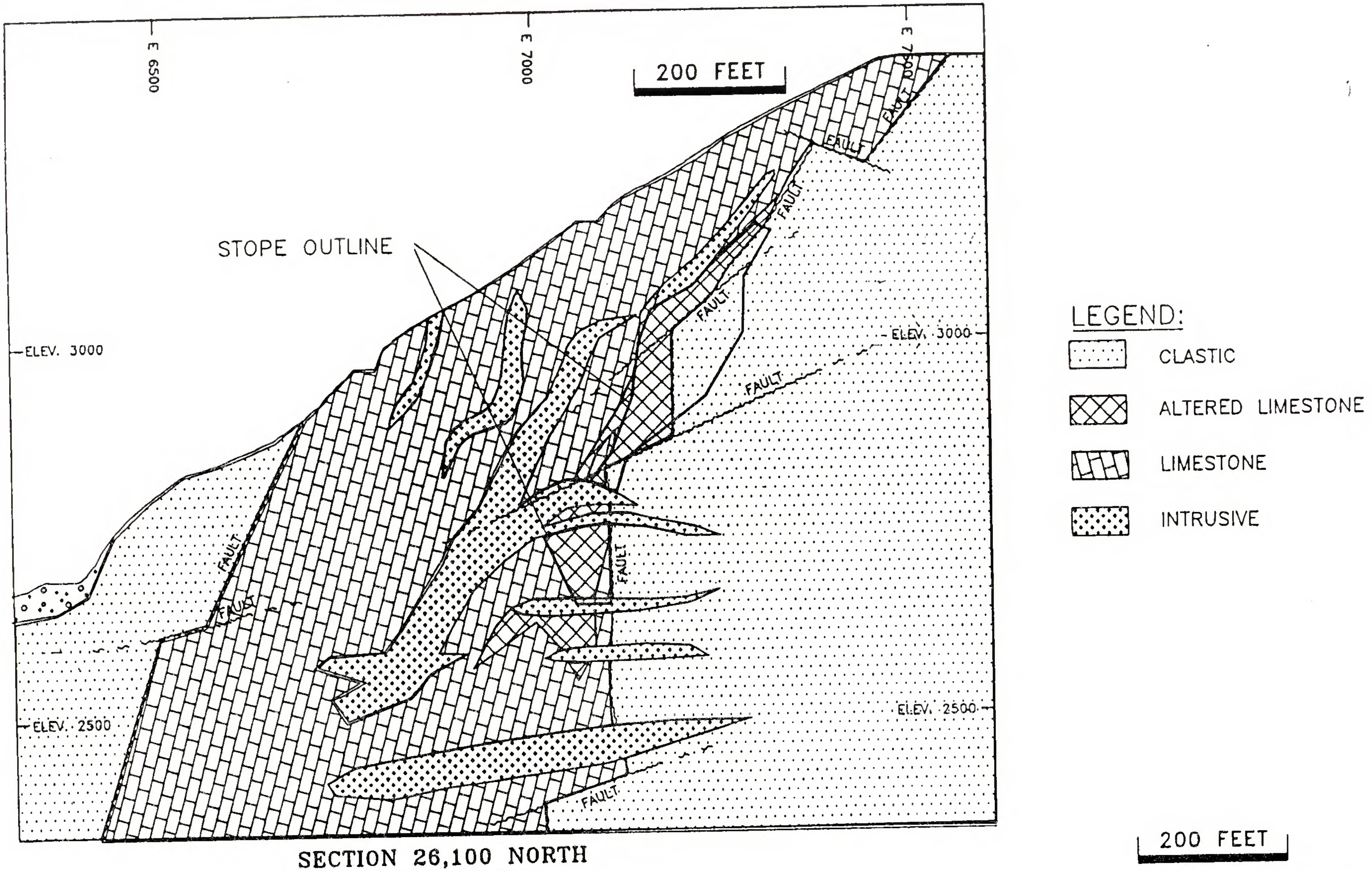
2.2.2 Waste Rock Volumes and Subsurface Lithologic Distribution

The relative proportions of the waste rock types were estimated based on the geologic logs and preliminary cross sections. A summary of the various lithologic proportions comprising the entire waste rock volume is illustrated in Figure 2-3. The limestone and clastic units comprise approximately 85 percent of the entire waste rock volume anticipated to be temporarily removed from the mine.

Data generated from surface core holes and the underground mapping were used to develop longitudinal sections of the proposed stoping areas in order to illustrate the relative distribution of lithologies. Figures 2-4 through 2-8 illustrate the relative abundance and location of the various rock types along the proposed stope walls. The west wall is dominated by low grade ore and unaltered limestone while the east wall is comprised almost entirely of clastic rock. The total surface area represented by the proposed stope walls is approximately 1.2 million square feet. The clastic, low grade ore, and unaltered limestone units comprise approximately 94 percent of the entire surface area.

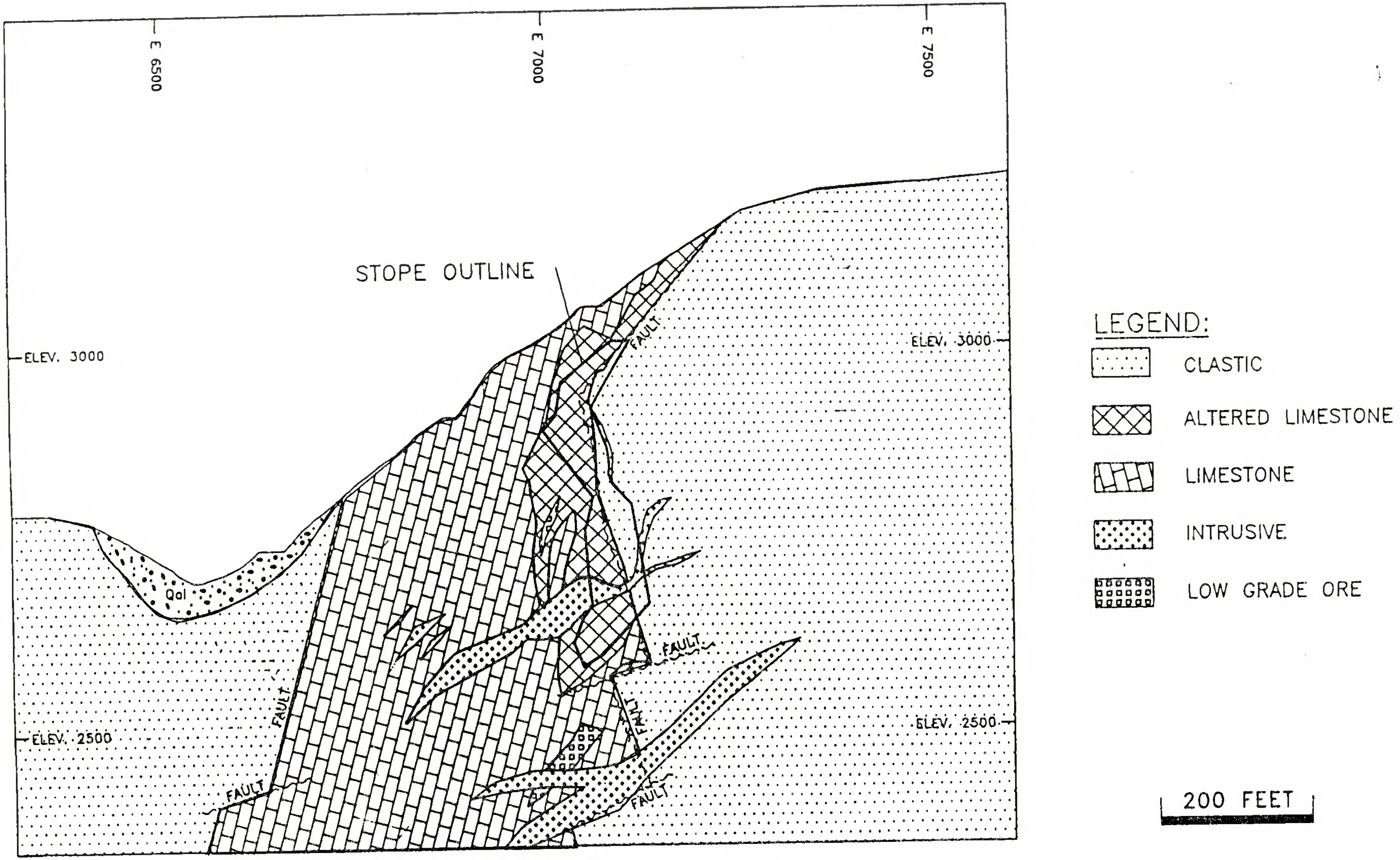
2.3 PHASE III - PRELIMINARY ACID BASE ACCOUNTING ANALYSES

To determine, in a preliminary manner, whether ARD/AMD would be a potential concern at the site, and to assess potential geochemical variability between the Key and Lamefoot lithologies, Echo Bay completed



SECTION 26,100 NORTH
LOOKING NORTH

CLIENT/PROJECT	ECHO BAY MINERALS Co. LAMEFOOT MINE KETTLE RIVER OPERATIONS REPUBLIC, WASHINGTON	Golder Associates		TITLE GEOLOGIC CROSS SECTION ILLUSTRATING THE DISTRIBUTION OF WASTE ROCK LITHOLOGIES
DRAWN	CHECKED J.C.	REVIEWED L.F.C.	DATE NOV 1993	SCALE AS SHOWN
FILE NO.	JOB NO. 933-2614	DWG NO./REV.NO. 261AGP	FIGURE 2-1	



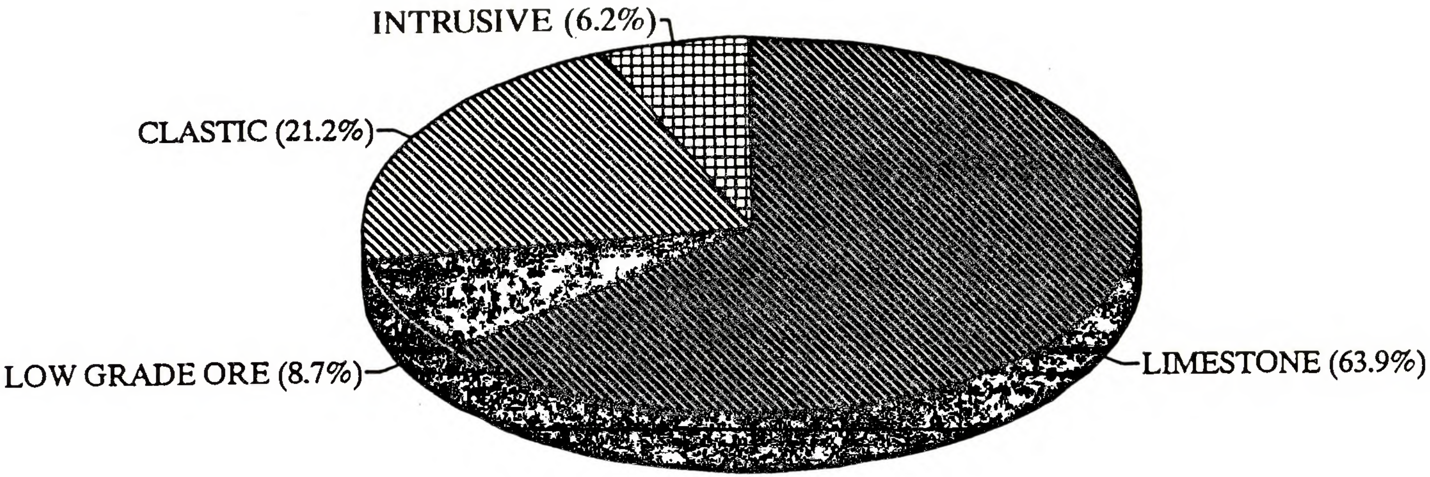
SECTION 26,900 NORTH
LOOKING NORTH

CLIENT/PROJECT
ECHO BAY MINERALS Co.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

Golder Associates

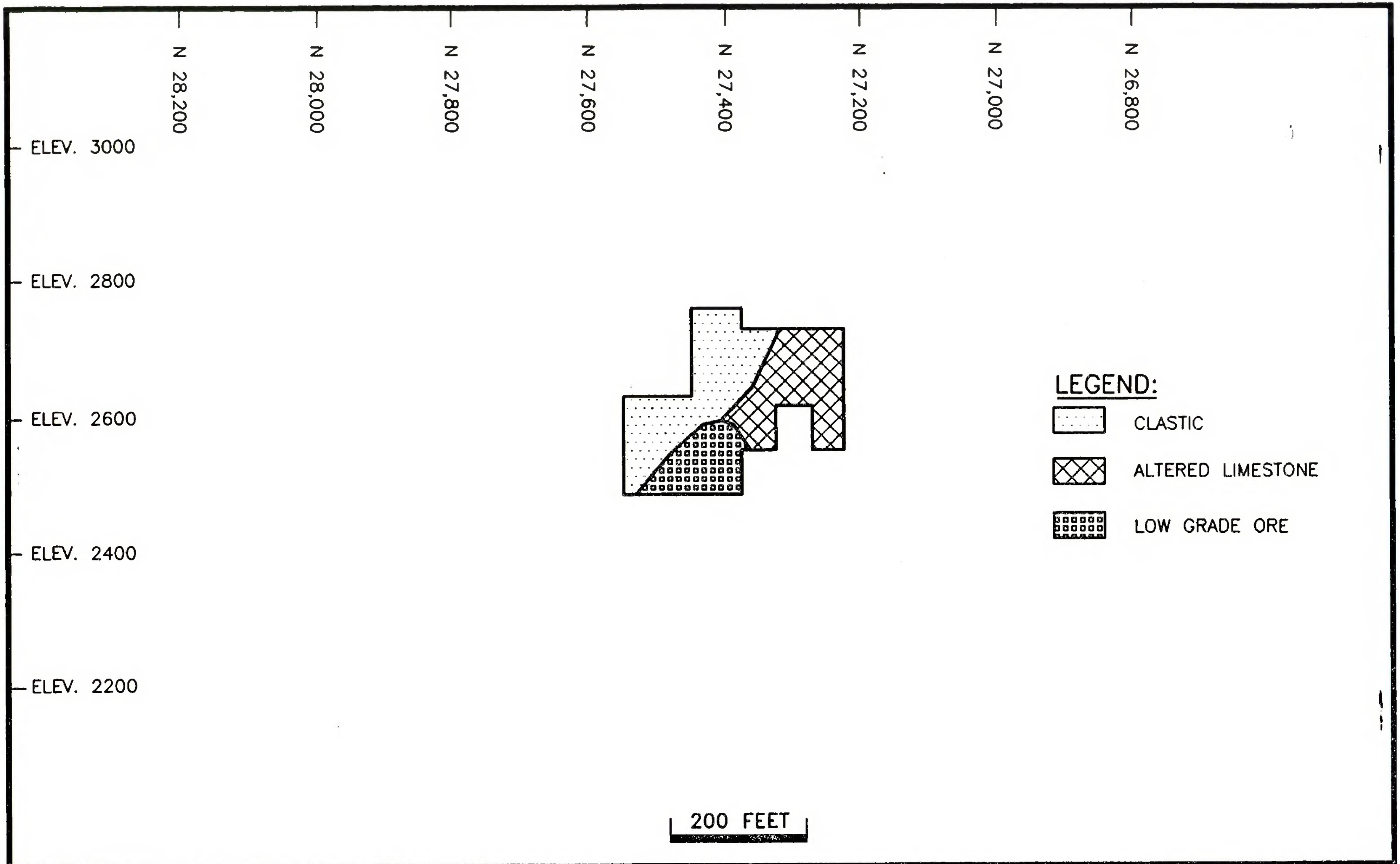
TITLE
GEOLOGIC CROSS SECTION ILLUSTRATING THE DISTRIBUTION OF WASTE ROCK LITHOLOGIES

DRAWN	CHECKED	REVIEWED	DATE	SCALE	FILE NO.	JOB NO.	DWG NO./REV.NO.	FIGURE
	J.C.	L.E.C.	NOV 1993	AS SHOWN		933-2614	269AGP	2-2



CLIENT/PROJECT	ECHO BAY MINERALS Co. LAMEFOOT MINE KETTLE RIVER OPERATIONS REPUBLIC, WASHINGTON		TITLE	ESTIMATED WASTE ROCK PERCENTAGES

DRAWN	CHECKED	REVIEWED	DATE	SCALE	FILE NO.	JOB NO.	DWG NO./REV.NO.	FIGURE
J.H.F.			NOVEMBER '93	AS SHOWN				2-3

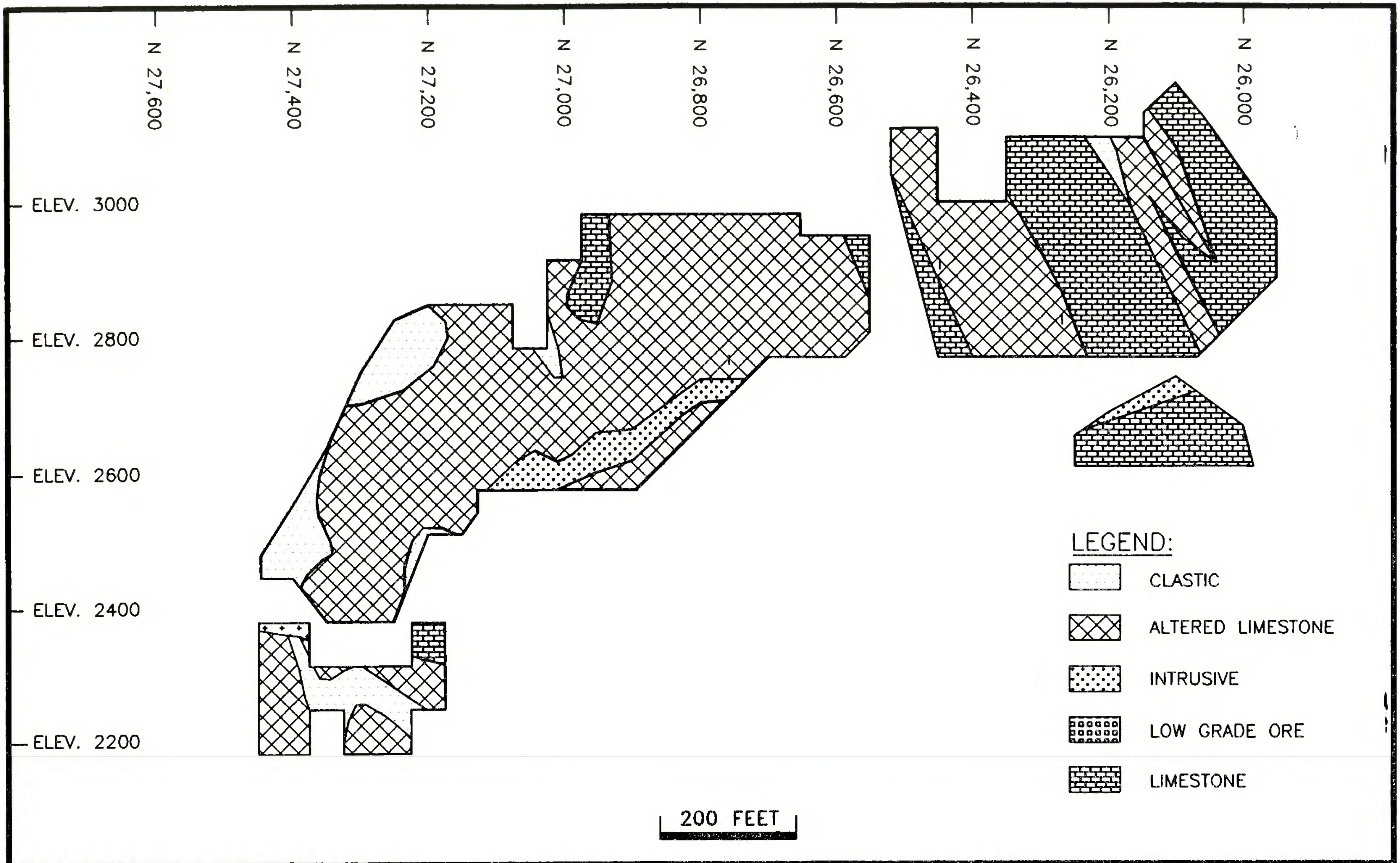


CLIENT/PROJECT
ECHO BAY MINERALS Co.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

Golder Associates
Denver, Colorado

TITLE
DISTRIBUTION OF WASTE ROCK LITHOLOGIES EAST WALL OF HW STOPE

DRAWN
CHECKED J.C.
REVIEWED L.E.C.
DATE AUGUST 1993
SCALE AS SHOWN
FILE NO.
JOB NO. 933-2614
DWG NO./REV.NO. A4
FIGURE 2-4

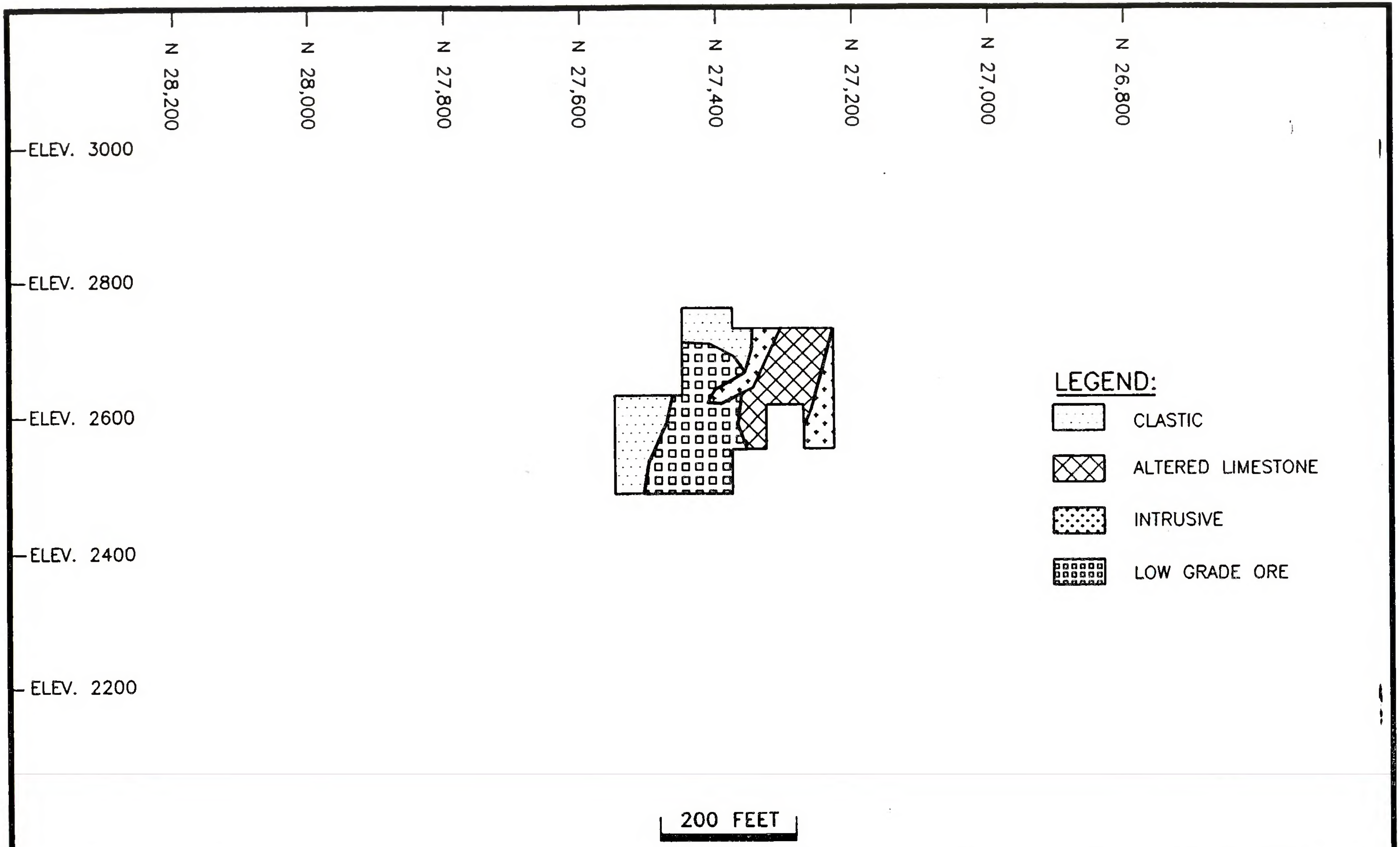


CLIENT/PROJECT
ECHO BAY MINERALS Co.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

Golder Associates
Denver, Colorado

TITLE
DISTRIBUTION OF WASTE ROCK LITHOLOGIES WEST WALL OF MAIN STOPES

DRAWN	CHECKED	REVIEWED	DATE	SCALE	FILE NO.	JOB NO.	DWG NO./REV.NO.	FIGURE
	J.C.	L.E.C.	AUGUST 1993	AS SHOWN		933-2614	A6	2-5

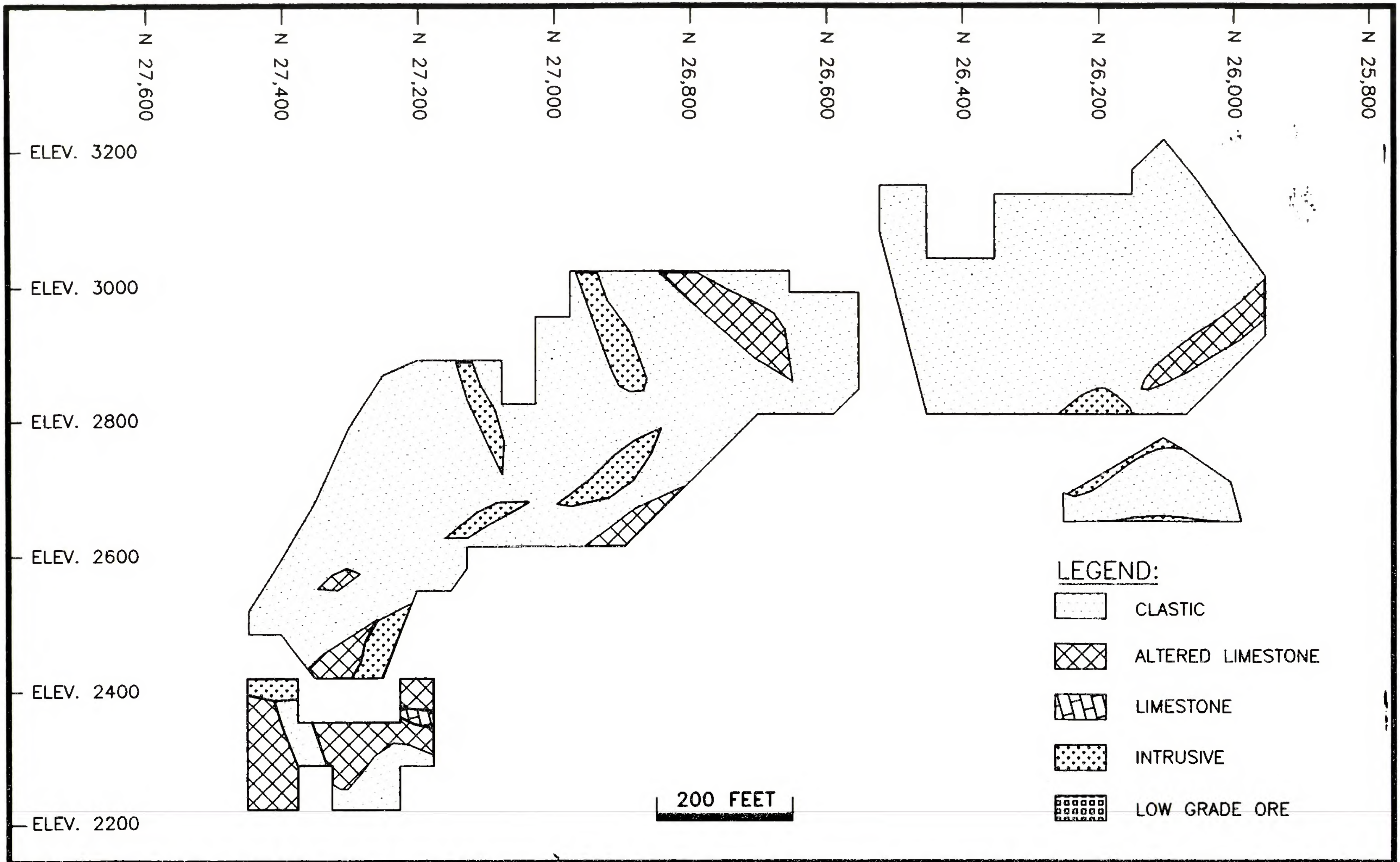


CLIENT/PROJECT
ECHO BAY MINERALS Co.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

Golder Associates
Denver, Colorado

TITLE
DISTRIBUTION OF WASTE ROCK LITHOLOGIES WEST WALL OF HW STOPES

DRAWN	CHECKED	REVIEWED	DATE	SCALE	FILE NO.	JOB NO.	DWG NO./REV.NO.	FIGURE
	J.C.	L.E.C.	AUGUST 1993	AS SHOWN		933-2614	A3	2-6

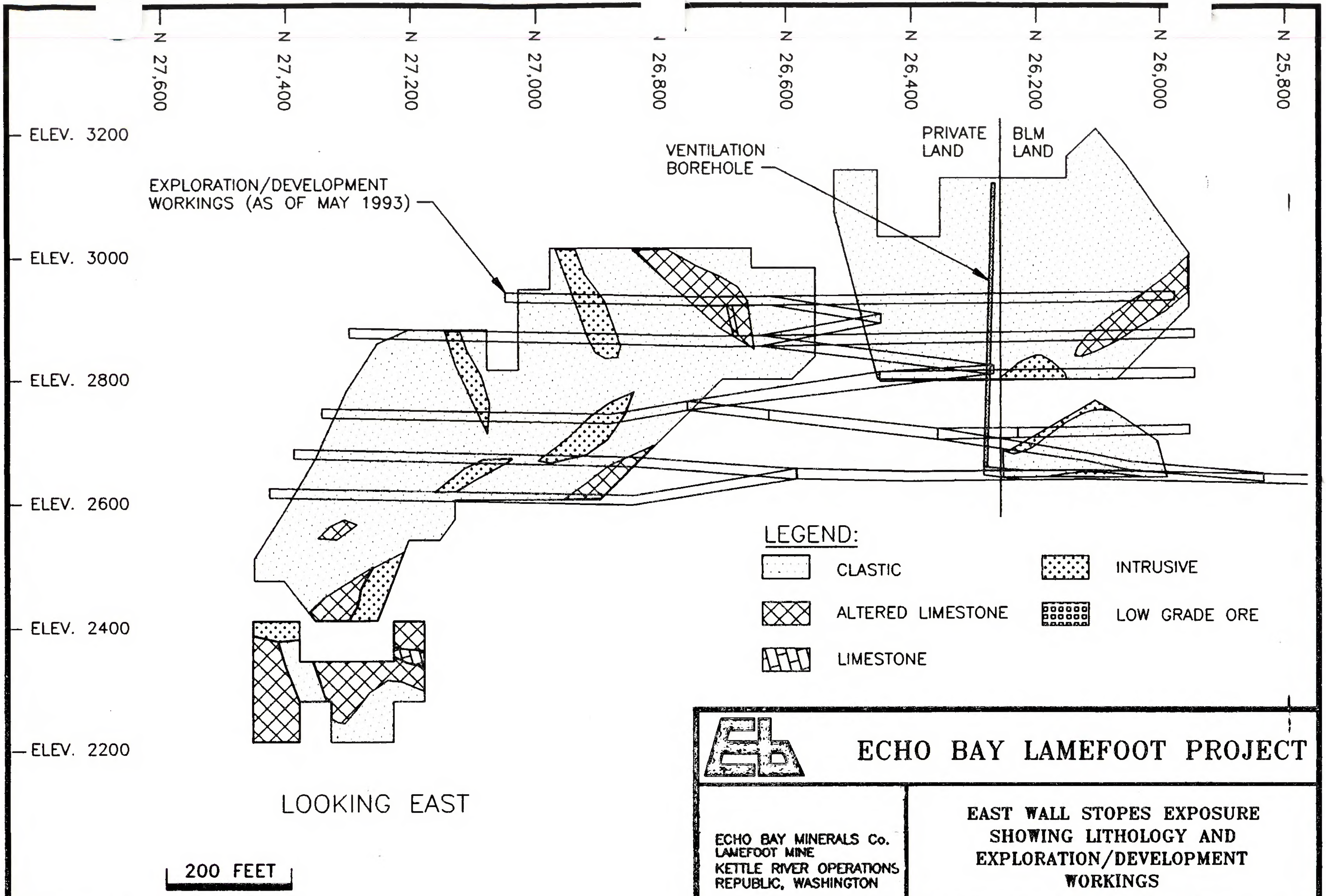


CLIENT/PROJECT
ECHO BAY MINERALS Co.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

Golder Associates
Denver, Colorado

TITLE
DISTRIBUTION OF WASTE ROCK
LITHOLOGIES EAST WALL
OF MAIN STOPES

DRAWN	CHECKED	REVIEWED	DATE	SCALE	FILE NO.	JOB NO.	DWG NO./REV.NO.	FIGURE
	J.C.	L.E.C.	AUGUST 1993	AS SHOWN		933-2614	A5	2-7



933-2614-A10

ECHO BAY LAMEFOOT PROJECT

ECHO BAY MINERALS Co.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

EAST WALL STOPES EXPOSURE SHOWING LITHOLOGY AND EXPLORATION/DEVELOPMENT WORKINGS

DATE	SCALE	FIG. NO. 2-8

acid base accounting tests on a number of waste rock samples in October, 1991. Tests were performed by Core Laboratories, located in Aurora, Colorado.

2.3.1 Sample Selection

The waste/mine wall rock samples selected for this preliminary testing satisfied the following criteria:

- The material will potentially report to the temporary waste rock placement area or be exposed in the stope walls of the underground mine.
- The degree of mineralization of a particular sample would, at a minimum, be representative of expected sulfide occurrence or provide a reasonably high bias to ensure an adequate degree of safety.
- The material was obtained from the area of the proposed stope or from rock which would report to the temporary waste rock placement area.

The methods used in sample selection consisted of:

- Evaluation of preliminary geologic cross-sections to determine the zones of ore and waste.
- Estimation of lithologic percentages comprising the waste rock material, based on examination of the geologic logs and preliminary cross-sections.
- Selection of core intervals and stope wall locations which reflected, to the greatest degree possible, the stratigraphic and spatial distribution within a given rock type.

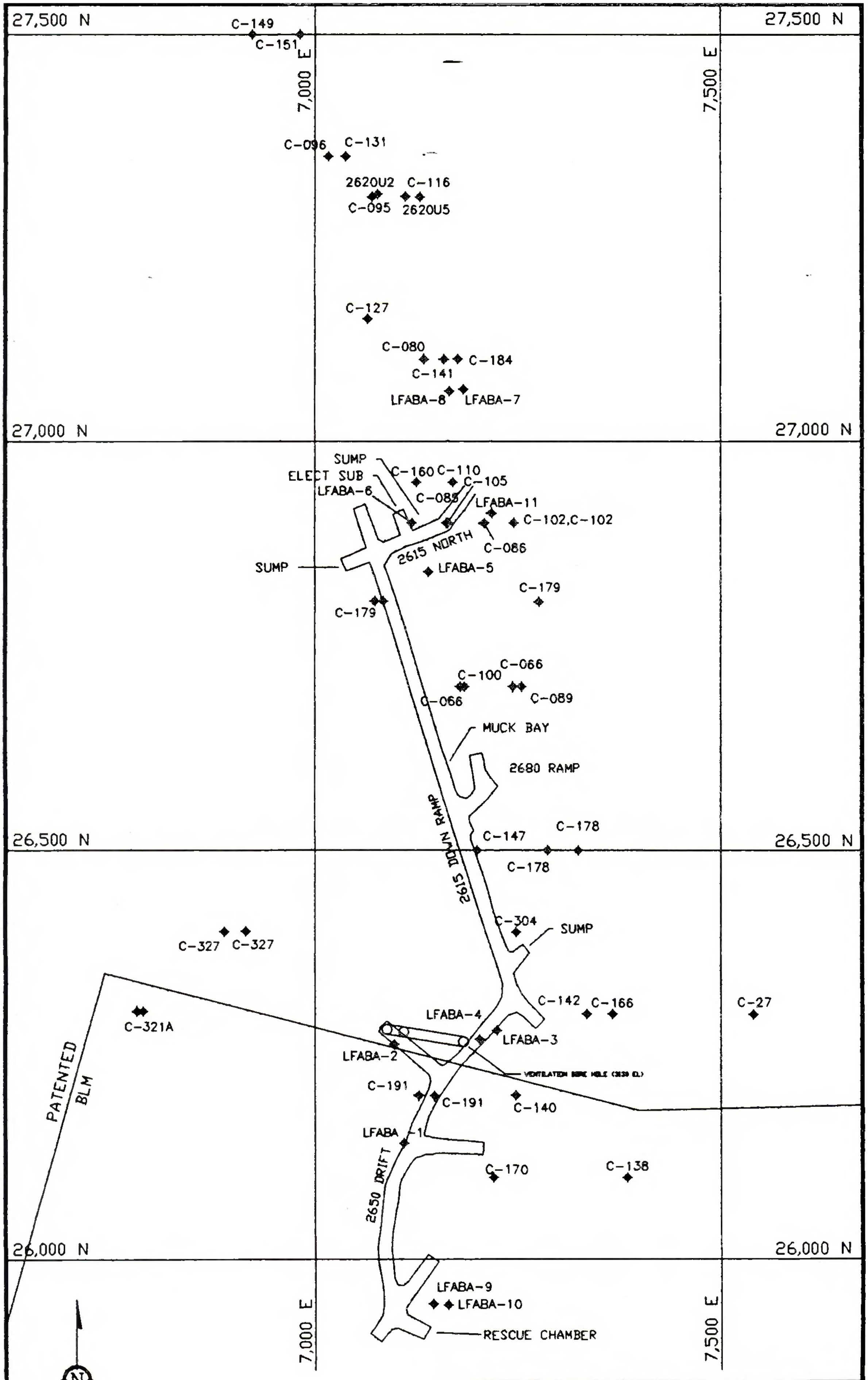
While every effort was made to satisfy the last point, the sampling of the stope walls and core intervals was spatially and mineralogically biased. The samples collected for the preliminary evaluation were collected at the stope boundary and as a result, are biased toward areas of increased mineralization. This was considered to be acceptable within the sampling program because the bias would integrate an additional factor of safety into the geochemical characterization. Therefore the samples submitted represent a "worst case" in terms of sulfide mineralization and acid generation potential.

2.3.2 Acid Base Accounting Procedures

A total of 11 waste/mine wall rock and 20 backfill samples were submitted for initial acid base accounting analysis (ABA). The waste/mine wall rock samples submitted for testing consisted of four clastic samples, one limestone sample, four altered limestone samples, and two intrusive samples. The sampling locations are illustrated on Figure 2-9. The geologic logs for the waste/mine wall rick and backfill samples submitted are presented in Tables 2-1 and 2-2, respectively. The following paragraphs summarize the test procedures, limitations and results for the waste rock and backfill acid base accounting.

Acid Base Accounting Procedures

ABA testing begins with a measurement of total sulfur in a sample, by nitric acid oxidation and subsequent titration (Sobek et al. 1978). The acid generation potential (AGP) of the waste material is then calculated from total sulfur content to tons $CaCO_3$/kiloton (TCaCO3/kT) of rock. The stoichiometric conversion used involves the oxidation of pyrite (FeS_2) and subsequent liberation and hydrolysis of iron (Fe) as illustrated by the reaction below [1].



ECHO BAY LAMEFOOT PROJECT

ECHO BAY MINERALS CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

WASTE AND MINE WALL ROCK GEOCHEMICAL SAMPLING LOCATION MAP

DATE: APRIL 1993 | SCALE: 1"=100' | FIG: 2-9

TABLE 2-1

SUMMARY OF WASTE/MINE WALL ROCK SAMPLES SUBMITTED FOR PHASE III ACID BASE ACCOUNTING ANALYSIS

LFABA Number	Rock Type	Sample Location			Geologic Description
		Northing	Easting	Elevation	
1	Low Grade Ore	26142	7110	2655	Altered limestone, trace hematite, trace pyrite, disseminated and in microfractures.
2	Intrusive	26263	7098	2655	Biotite rhyodacite, propylitically altered, trace carbonate mineralization occurring in veinlets.
3	Clastic	26280	7224	2650	Clastic, silicic, trace pyrite in veinlets, and disseminations, trace carbonate mineralization.
4	Intrusive	26269	7203	2722	Biotite rhyodacite, minimal alteration, trace pyrite occurring in veinlets.
5	Low Grade Ore	26840	7140	2755	Altered limestone, trace hematite, trace magnetite, trace calcite, fractured.
6	Low Grade Ore	26900	7120	2615	Massive hematite and magnetite, trace carbonate mineralization, 2% pyrite in veinlets.
7	Clastic	27063	7184	2620	Clastic, silicic, 2-5% pyrite in veinlets and microfractures, some disseminations.
8	Low Grade Ore	27061	7166	2620	Altered limestone, 60% massive hematite and magnetite, trace sulfide mineralization.
9	Clastic	25945	7146	2730	Clastic, sharpstone, quartz veining with trace pyrite disseminated throughout.
10	Limestone	25944	7165	2730	Limestone, unaltered, weak microfracturing, trace sulfide mineralization consisting of disseminated pyrite.
11	Clastic	26912	7218	2745	Clastic, siltstone/sharpstone, minor quartz veining, 2-5% pyrite in veinlets and as disseminations.

TABLE 2-2

GEOLOGIC LOGS FOR BACKFILL EXPLORATION HOLES

Borehole Name	Depth (fbgs*)	Geologic Description
LR-1	0-10'	Overburden
	10-50'	Intrusive (Scatter Creek)
		- fine grained, medium green
		- local small biotite phenocrysts
		Possible coarse, psdomorphs of feldspar phenocrysts
		- trace calcite
		- local trace fine disseminated pyrite
	50'	End of hole
LR-2	0-5'	Overburden
	5-60'	Intrusive, medium green, fine grained with occasional biotite phenocrysts
		- minor irregular plagioclaise phenocrysts
		- minor carbonate (calcite)
	60-80'	- local trace disseminated pyrite
		Clastic, fine grained, mixed oxidized material with very fine grained
	80'	siliceous (chert) pyritic rock
		Similar to 60-80' except for increased very fine grained siliceous material.
		- weakly to moderately pyritic
	110-130'	- local moderate oxidation - presumably from sulfides
		Increased amount of oxidized material possibly masking pyritic siliceous rock (very little apparent).
	130-180'	Similar to 80-110'; mixed gray clastic
		- fine grained but less siliceous and pyritic with oxidized fine grained rock
		- clastics exhibit a subtle "blocky" texture due to parting on foliation (bedding) planes and a probable cross cutting fabric
	180-220'	Decreased oxidized rock, predominary light gray clastic, weakly siliceous
		- strongly pyritic w/abundant disseminated pyrite (1-2%)
		- sulfide occurrence appears to be locally controlled by cross cutting thin silica veinlets
	220'	- continued subtle blocky texture described above
		End of hole

$$FeS_2 + 15/4\ O_2 + 7/2\ H_2O \rightarrow Fe(OH)_3 + 2SO_4^{-2} + 4H^+ \qquad [1]$$

Total sulfur has been used for this analysis rather than pyritic sulfur because there are no other sulfur bearing minerals (i.e., barite, jarosite or alunite) found in significant amounts within the waste rock which could result in positive interference. Using total sulfur ensures that an additional factor of safety is incorporated into the method of evaluation. In some instances the mineralogy of the waste rock is such that this is an overly conservative factor of safety. However, based on the current understanding of the Lamefoot mineralogy, using total sulfur to evaluate acid generation potential provides an acceptable level of safety and is consistent with the method of evaluation used in the Key Expansion program.

The acid neutralizing potential (ANP) is a function of reactions which liberate hydroxyl (OH-) and carbonate (HCO_3^- and CO^{-2}) ions. These reactions include the dissolution of carbonates [2] and desorption of alkali cations and breakdown of silicate minerals to clays [3].

$$4H^+ + 2CaCO_3 \rightarrow 2Ca^+ + 2CO_2 + 2H_2O \qquad [2]$$

$$CaAl_2Si_2O_8 + 2H^+ + H_2O \rightarrow Al_2Si_2O_5(OH)_4 + Ca^{+2} \qquad [3]$$

The ANP is determined by measuring experimentally, the quantity of acid consumed by a finely ground sample of waste material. The total acid consumed is then converted to tons $CaCO_3$/kT waste material.

The Net Neutralization Potential (NNP) of a waste material is subsequently calculating using the following equation [4]:

$$NNP = ANP\text{-}AGP \qquad [4]$$

A negative NNP value indicates that the waste material is potentially acid producing while a positive NNP value indicates an excess of neutralization potential. Because the NNP is expressed in $CaCO_3$ equivalents, the value represents the amount of pure (100% $CaCO_3$) that would be necessary to add to 1000 tons of waste material to neutralize all acidity produced.

2.3.3 Interpretation of ABA Results

An ABA test determines the total amount of sulfur in all forms present in a material, including sulfide-sulfur. In calculating the AGP, all the sulfide present is assumed to be pyrite. This is a valid assumption for the sulfide mineralization present at Lamefoot. The extensive geologic data indicate that pyrite is the dominant sulfide mineral present with minor amounts of pyrrhotite and chalcopyrite. The maximum AGP is determined assuming all pyrite will weather and produce acid by reaction [1]. Similarly, the ANP determines the amount of acid a material can neutralize [reactions 2 and 3] by measuring the acidity consumed during a titration procedure. The NNP is then calculated by subtracting the AGP from the ANP.

Given the potential ambiguity and somewhat subjective nature of interpretation of ABA data when used to determine the potential for acid generation in field conditions, several criteria have been developed to predict if a sample will produce acid in the field. In one interpretation, a NNP value of <-20 T $CaCO_3$/kT represents a net acid-producing material (Group 1) while a NNP value of >20 T $CaCO_3$/kT represents a net neutralizing material (Group 3). NNP values between -20 and 20 T $CaCO_3$/kT suggest an uncertain potential for acid generation (Group 2) (B.C. Acid Rock Drainage Task Force 1989).

Another method for interpretation of acid base accounting is the evaluation of the ANP:AGP ratio. Based on experience with laboratory tests and field data on a wide range of sulfur bearing rock types, there appears to be a low potential for acid generation for waste rock samples with an ANP:AGP ratio greater than or equal to 3:1 (Group 3). Waste rock samples with a ratio of less than 1:1 (Group 1) have an increased potential to generate acid. Those waste rock sampled with ANP:AGP ratios between 1:1 and 3:1 (Group 2) have an acid generation potential which is difficult to predict (Brodie et al. 1991; and, Smith 1992). These three groups were developed to conservatively represent responses expected in the field given the inherent inaccuracies in extrapolating acid base accounting results to field scale situations.

The method used to interpret ABA results for the Lamefoot samples consisted of dividing the samples into three categories based on their NNP and ANP:AGP values. As mentioned above, all AGP values were determined assuming the total sulfur contained within a sample to be pyritic sulfur. Table 2-3 summarizes the criteria used to evaluate the Lamefoot ABA data.

2.3.4 Limitations

Although the ABA test has some shortcomings, it provides one of the best, most cost-effective approaches to initial evaluation of geochemical behavior. ABA test results are generally considered conservative because they assume the total sulfur content of a material (S-total) to be sulfide-sulfur (S-pyrite).

The actual geochemical behavior of the waste materials is not considered by static testing (i.e., ABA tests) which provides a total mass balance of acid generating and neutralizing materials. Following the discussion above, it is necessary to consider environmental factors that are not addressed in the testing.

Although, the ABA values for a material may indicate that there is a strong likelihood for the production of acid associated with the weathering of a material, several factors will serve to decrease the actual amount of acid produced compared to the amount predicted by the NNP. Acid producing potentials determined from ABA data are conservative in that the sample grain size used for the analysis is minus 60 (or -200) mesh compared to the larger size fractions (generally 1/4 to 6 inches or larger) for typical mined ore and waste material. In a large particle, much of the pyrite will be bound within the rock matrix and unavailable for oxidation. As the size fraction decreases, the effective surface area increases and hence, the reactivity of a given sample increases. Thus, the surface area and reaction rate associated with the laboratory sample are much greater than that of the actual on site material in the field.

The ABA test also assumes that all pyritic sulfur will weather and all iron produced will ultimately undergo hydrolysis and form $Fe(OH)_3$. This results in a maximum production of 4 moles of H^+ for every mole of pyrite weathered [Reaction 1]. In a natural system, it is unlikely that all pyrite will react or that all the iron liberated will precipitate as $Fe(OH)_3$. In addition, iron will form other sulfate and carbonate minerals, depending on pH, and may also co-precipitate in these mineral forms with other metals, rendering the iron unavailable for hydrolysis and the subsequent formation of $Fe(OH)_3$.

In addition to unreactive pyrite contained within the rock matrix, some pyrite will not weather due to the precipitation of iron hydroxides on the surface of the pyrite crystals, rendering them non-reactive. This is especially true in a low flux environment where the reaction products are not subject to transport. Likewise, it is important to note that the acid generating potential can be under estimated due to the precipitation of iron hydroxide, and potentially gypsum, on the surfaces of acid neutralizing minerals.

The tendency for a material to produce acid is also strongly site specific and determined by factors including climate, associated lithologic and sulfide mineralogy, design and mitigative measures, and soil and rock buffering capacity. Design measures such as placing potential acid-producing materials above or within potential acid-neutralizing materials have the capacity to produce neutral drainage.

Soil systems are inherently moderate to well buffered. Thus, all H^+ ions produced will not necessarily result in a decrease in solution pH. Some H^+ ions will be involved in precipitation and dissolution reactions with metals and other ions in solution and on exchange sites, decreasing the amount of H^+ in solution. Although there may be a limited neutralization capacity, there may be a considerable buffering capacity.

TABLE 2-2
(Concluded)

Borehole Name	Depth (fbgs*)	Geologic Description
LR-3	0-5'	Overburden
	5-110'	Primarily dark gray to brownish fine grained clastic - possible siltstone. - minor oxidation - generally more prevalent in the uphole portions - little to no sulfide visible - local minor light colored, more siliceous clastics - possible thin interbeds in the siltstones; the lighter colored clastics commonly are very weakly pyritic
	110-130'	Predominantly lighter gray relative to uphole; increased silica with sulfides. - subtle blocky texture probably due to clastics parting on bedding planes - trace silica veining
	130'	End of hole

*fbgs - Feet below ground surface

TABLE 2-3

SELECTED EVALUATION CRITERIA FOR THE LAMEFOOT ACID-BASE ACCOUNTING ANALYSES

	NNP	ANP:AGP
Strong Acid Generation Potential		
Group 1	NNP $<$-20 $TCaCO_3/kT$	ANP:AGP $<$ 1.0
Uncertain Acid Generation Potential		
Group 2	-20 $TCaCO_3/kT \leq$ NNP $\leq$ 20 $TCaCO_3/kT$	1.0 $\leq$ ANP:AGP $\leq$ 3.0
Low Acid Generation Potential		
Group 3	NNP $>$ 20 $TCaCO_3/kT$	ANP:AGP $>$ 3.0

TABLE 2-4

RESULTS OF PHASE III ACID-BASE ACCOUNTING ANALYSIS OF WASTE/MINE WALL ROCK SAMPLES

Sample Name	Sample Lithology	Total Sulfur (%)	AGP $TCaCO_3/kT$	ANP $TCaCO_3/kT$	NNP $TCaCO_3/kT$	ANP:AGP Ratio
LFABA #1	Low Grade Ore	0.02	0.6	475	474.4	791.6
LFABA #2	Intrusive	1.00	31.2	332	300.8	10.6
LFABA #3	Clastic	2.79	87.2	51.6	-35.6	0.59
LFABA #4	Intrusive	0.42	13.1	177	163.9	13.51
LFABA #5	Low Grade Ore	0.01	0.3	439	438.7	1.463
LFABA #6	Low Grade Ore	1.65	51.6	434	382	8.41
LFABA #7	Clastic	0.72	22.5	72.8	50.3	3.23
LFABA #8	Low Grade Ore	0.82	25.6	210	184.4	8.20
LFABA #9	Clastic	3.08	96.2	68.0	-28.2	0.70
LFABA #10	Limestone	1.89	59.1	464	404.9	7.85
LFABA #11	Clastic	7.03	220	61.9	-158.1	0.28

TABLE 2-5

RESULTS OF PHASE III ACID BASE ACCOUNTING OF LAMEFOOT BACKFILL SAMPLES

Core Hole	Depth Interval (fbgs)[1]	Totals %	AGP $TCaCO_3$/kT	ANP $TCaCO_3$/Kt	NNP $TCaCO_3$/kT	ANP:AGP
LR-1	0-20	0.096	3	37	34	12.3
	20-40	<0.01	<0.01	67	67	>67
	40-50	<0.01	<0.01	66	66	>66
LR-2	0-20	<0.01	<0.01	51	51	>51
	20-40	<0.01	<0.01	45	45	>45
	40-60	0.22	7	61	54	7.7
	60-80	0.51	16	43	27	2.7
	80-100	1.44	45	44	-1	0.98
	100-120	2.81	88	43	-45	0.48
	120-140	0.70	22	20	-2	0.90
	140-160	1.95	61	67	6	1.09
	160-180	0.29	9	25	10	1.56
	180-200	0.93	29	71	42	2.45
	200-220	0.99	31	81	50	2.61
LR-3	0-20	0.25	8	33	25	4.12
	20-40	0.54	17	47	30	2.76
	40-60	0.93	29	69	40	2.37
	60-80	0.32	10	53	43	5.3
	80-100	0.38	12	50	38	4.16
	100-120	0.42	13	50	37	3.84

Note: [1]fbgs = Feet Below Ground Surface

Finally, the ABA test does not consider the kinetics of acid producing or neutralization reactions. Thus, a material may appear to be acid producing based on ABA results, but the rate of acid producing reactions and may be slower than neutralizing reactions. Likewise, an apparently neutral material may produce acid if neutralization reactions are kinetically limited, particularly if the amount of pyritic-S is significant (>1%). Therefore, the usefulness of ABA data is only in a descriptive context and not predictive unless combined with kinetic testing. Kinetic testing confirms the accuracy of the criteria established to evaluate ABA data in predicting potential geochemical behavior. The kinetic testing performed during the Key Expansion waste characterization program confirmed that the ABA data for the waste rock lithologies could be safely used in a predictive context. In fact, the criteria established to evaluate the ABA data for the Key Expansion (and hence the Lamefoot Project) provided a conservative framework appropriate for minimizing, to the largest degree, potential environmental impacts.

2.4 RESULTS

The results of the acid base accounting analyses for the 11 initial waste/wall rock samples and 20 potential backfill samples are presented in Tables 2-4 and 2-5, respectively. The paragraphs which follow discuss the results for each of the waste/wall rock lithologies at Lamefoot. In addition, these results are compared to the ABA results for the equivalent Key Expansion waste rock types in order to illustrate potential geochemical similarities and differences.

Clastic

Total sulfur values of the four Lamefoot clastic samples ranged from 0.72 to 7.03 percent, with a geometric mean of approximately 2.57 percent. Each of the samples has significant amounts of neutralization capacity with ANP values ranging from 51.6 to approximately 73 $TCaCO_3/kT$, and a geometric mean of 63 $TCaCO_3/kT$. The NNP values for the clastic samples range from -158.1 to 50.3 $TCaCO_3/kT$, while the ANP:AGP ratios range from 0.28 to 3.23 with a geometric mean of 0.78. Three samples (LEAG-3, LFABA-9, and LFABA-11) had NNP values less than -20 $TCaCO_3/kT$ and ANP:AGP ratios less than 1 suggesting a strong acid generation potential. These samples are subsequently classified as Group 1 materials. The remaining sample (LFABA-7) is classified as Group 3 with an NNP value greater than 20 $TCaCO_3/kT$ and a ANP:AGP ratio greater than 3.0.

The range of total sulfur values for the Lamefoot clastic samples is comparable to that of the Key Expansion clastic samples. Total sulfur values for the Key Expansion clastic samples range from less than 0.01 to 3.06 percent with a mean of 0.32 percent. ANP values for the Key Expansion clastic samples range from 27.7 to 217 $TCaCO_3/kT$ and have a geometric mean of 45.5 $TCaCO_3/kT$. Therefore, the preliminary acid base accounting data suggest that the selected Lamefoot clastic samples contain slightly higher total sulfur and greater neutralizing capacity when compared to the Key Expansion clastic samples. Additional clastic samples have been submitted for acid base accounting in order to characterize the degree of variability within the clastic unit present at the Lamefoot Project. A discussion of the additional testing is presented in Section 3.0.

Limestone

One limestone sample was submitted for acid base accounting (LFABA-10). The total sulfur is approximately 1.9 percent, while the NNP and ANP:AGP values are 404.9 $TCaCO_3/kT$ and 7.85, respectively. The limestone present in wall rock at Lamefoot is subsequently classified as a Group 3 material, with a negligible potential for acid generation and a significantly high acid neutralization capacity. The mean total sulfur content for the Key Expansion limestone is approximately 0.1 percent, suggesting that the selected limestone sample from the Lamefoot site contain a higher percentage of sulfide mineralization. Because of the high degree of acid neutralizing capacity, it is not anticipated that this is a significant amount of sulfide mineralization relative to expected geochemical behavior. Additional samples have been submitted for acid base accounting in order to characterize the potential variability in the limestone present at the Lamefoot site. A discussion of the additional testing is presented in Section 3.0.

Low Grade Ore

The total sulfur values of the four low grade ore samples submitted for acid base accounting range from 0.01 to 1.65 percent with a geometric mean of 0.13 percent. NNP values range from 184 to 474 $TCaCO_3/kT$ while ANP:AGP ratios range from 8.20 to 1,463 indicating that the low grade ore, which is composed of altered limestone, has a negligible potential to generate acid and a significantly high neutralization potential (Group 3).

Intrusive

Total sulfur values for the two intrusive samples submitted for acid base accounting range from 0.42 to 1.0 percent, with a geometric mean of 0.65 percent. NNP values range from 164 to approximately 301 $TCaCO_3/kT$, while ANP:AGP values range from 10.6 to 13.5. This indicates that the intrusive waste rock has no significant potential for acid generation (Group 3) and the neutralizing capacity is in excess of the maximum potential acidity. The intrusive waste rock at the Key Expansion exhibited the same characteristics relative to the potential for acid generation. Total sulfur values for the Key Expansion intrusives range from less than 0.01 to 0.07 percent with a geometric mean of 0.013 percent. ANP:AGP ratios for the Key Expansion intrusive waste rock range from 35.7 to 303.3 suggesting that the selected Lamefoot intrusive samples have slightly higher total sulfur contents and slightly lower acid neutralizing potential.

Backfill

Total sulfur values for the potential backfill range from less than 0.01 to 2.81 percent with a geometric mean of approximately 0.25 percent. NNP values range from -45 to 67 $TCaCO_3/kT$ while ANP:AGP values range from 0.48 to greater than 67.0. Of the 20 samples submitted three are classified as Group 1, seven as Group 2 and 10 as Group 3. The results of the acid base accounting analysis for the backfill samples are summarized on Table 2-4. Based on the geologic descriptions and acid base accounting data it appears that the clastic rocks comprising the backfill correlate to the Clastic I domain present at the Key Expansion.

2.5 SUMMARY OF PHASE III PRELIMINARY ACID BASE ACCOUNTING

2.5.1 Waste/Mine Wall Rock Samples

Eleven waste/mine wall rock samples were submitted for preliminary acid base accounting. Table 2-6 provides a summary of the results relative to the criteria defined in Section 2.3.3. The laboratory results appear in Appendix A. Three of the eleven samples were subsequently classified as Group 1 materials with a strong potential to generate acid. All of these Group 1 samples were from the clastic lithology. The limestone, low grade ore, and intrusive samples are all classified as Group 3 materials and show negligible acid generation potential and significant amounts of neutralization capacity. No Group 2 samples were identified through the preliminary analysis. The Group 3 lithologies represent approximately 79 percent of the total waste rock volume and 54 percent of the expected mine wall exposure area. The clastic lithology comprises approximately 21 percent of the total waste rock volume and 46 percent of the expected mine wall exposure area. Given the magnitude of the neutralizing capacity of the remaining 79 percent of the waste rock, it is not anticipated that the temporary waste stockpiles will represent a significant source of potential acid generation. The continuing waste characterization work (Section 3.0) focused on defining what portion of the clastic rock has the highest potential for acid generation (i.e., develop additional waste rock domains based on geochemical homogeneity). In addition, subsequent phases were designed to emphasize the characterization of the AMD potential from the mine wall rock.

TABLE 2-6

ABA RESULTS RELATIVE FOR THE WASTE/MINE WALL ROCK SAMPLES RELATIVE TO THE EVALUATION CRITERIA

Waste/Mine Wall Rock Lithology	Number of Samples Submitted	Evaluation Criteria NNP	 ANP:AGP
		Geochemical Classification	
Limestone	1	(1) Group 3	(1) Group 3
Low Grade Ore	4	(4) Group 3	(4) Group 3
Clastic	4	(3) Group 1 (1) Group 3	(3) Group 1 (1) Group 3
Intrusive	2	(2) Group 3	(2) Group 3
TOTAL	11		

Notes: NNP reported as $TCaCO_3/kT$
( ) refers to the number of samples in a particular group

In general, the total sulfur values of the 11 samples collected from the various rock lithologies were higher than those found at the Key Expansion site. As mentioned above, the samples collected for the preliminary evaluation were collected at the stope boundary and as a result, are biased toward areas of increased mineralization. This was considered to be acceptable within the sampling program because the bias would integrate an additional factor of safety into the waste characterization. The samples submitted therefore represent a "worst case" in terms of sulfide mineralization and acid generation potential. Several additional factors which result in establishing a high degree of conservativeness include the following:

- Use of total sulfur values to calculate the AGP effectively overestimates the maximum potential acidity of a particular sample.
- Assuming that all sulfide minerals present will be available for reaction and the oxidation of the sulfide minerals will go to completion.
- Procedural characteristics of the acid base analytical method over-estimate the magnitude of sulfide oxidation by increasing the effective surface area and using a more aggressive oxidizing agent than found in natural systems.

As a result of the initial acid base accounting a significant amount of variability within the clastic domain was identified. In addition, this lithology has the highest potential for acid generation of the four waste rock types. In order to characterize the variability and to further define potential geochemical domains within this unit, additional samples were selected and submitted for acid base accounting analysis as part of Phase IV and V of the geochemical characterization. In addition, as part of the continuing waste characterization program developed for the Lamefoot Project, additional samples from the limestone, altered limestone, low grade ore, and intrusive waste rock lithologies were submitted for testing.

2.5.2 Backfill Samples

Twenty samples collected from the potential backfill were submitted for acid base accounting analysis. Table 2-7 provides a summary of the results relative to the established evaluation criteria. Based on the ANP:AGP criteria, three of the samples are considered to have a strong potential for acid generation (Group 1), seven of the samples have an uncertain acid generation potential (Group 2), and 10 of the samples have negligible acid generation potential and significant neutralization potential. When evaluated relative to the NNP criteria, one sample has a strong acid generation potential (Group 1), four samples have uncertain acid generation potential (Group 2), and 15 samples have significant neutralization potential. A comparison of the geologic logs for the Clastic I lithology at the Key Expansion with those clastic descriptions from the backfill exploratory boreholes suggest a distinct similarity. The acid base accounting data indicate comparable sulfur contents and neutralizing capacity, further supporting the similarity of the clastic rocks between the two sites.

Variability of results can be accounted for by weathering and vertical distribution of oxidation. These characteristics are noted on the geologic logs (Table 2-2) and as a result identification of these zones on a visual basis is facilitated making identification in the field possible.

3.0 CONTINUING WASTE CHARACTERIZATION PROGRAM

Beginning in 1993, the second portion of the phase waste characterization program was implemented and consists of the following elements:

- Phase IV - Selection of Additional Mine Wall Rock Samples;
- Phase V - Secondary Acid Base Accounting Laboratory Analysis;

TABLE 2-7

ABA RESULTS FOR THE BACKFILL SAMPLES RELATIVE TO THE EVALUATION CRITERIA

Geochemical Classification	Evaluation Criteria	
	NNP	ANP:AGP
Group 1	LR-2 (100-120)	LR-2 (80-100) LR-2 (100-120) LR-2 (120-140)
Group 2	LR-2 (80-100) LR-2 (120-140) LR-2 (140-160) LR-2 (160-180)	LR-2 (60-80) LR-2 (140-160) LR-2 (160-180) LR-2 (180-200) LR-2 (200-220) LR-3 (20-40) LR-3 (40-60)
Group 3	LR-1 (0-20) LR-1 (20-40) LR-1 (40-50) LR-2 (0-20) LR-2 (20-40) LR-2 (40-60) LR-2 (60-80) LR-2 (180-200) LR-2 (200-220) All LR-3 Samples	All LR-1 Samples LR-2 (0-20) LR-2 (20-40) LR-2 (40-60) LR-3 (0-20) LR-3 (60-80) LR-3 (80-100) LR-3 (100-120)

- Phase VI - Definition of Geochemical Mine Wall Rock Domains; and,
- Phase VII - Geochemical Correlation with Key Expansion.

A brief discussion of each element of the waste characterization program is presented below.

3.1 PHASE IV - SELECTION OF ADDITIONAL MINE WALL ROCK SAMPLES

A total of 30 samples were selected from rock core drilled within the proposed Lamefoot stoping areas. Collar plots illustrating the location of the core holes selected for sampling are presented in Figure 2-9.

Within a given lithology, core samples were selected to represent the largest stratigraphic and spatial distribution possible. However, because of the configuration of the ore body, the majority of the core samples are biased spatially (and hence geochemically) toward the mineralized zone. Therefore, the sampling program developed for the selection of additional samples is not only consistent with the preliminary program but also ensures that the present program is adequately conservative.

Samples were **not composited** across drill holes or along wide elevation differences within the same hole in order to avoid obscuring trends with respect to acid generation potential. Samples were collected over an average elevation difference of approximately 6 feet within any given lithology. For the purposes of geochemical testing, this level of compositing provided representative, discrete, point samples of each rock type.

The number of samples chosen from a particular lithologic domain correlates directly to; (1) the potential for geochemical variability and acid generation potential (as defined through the preliminary testing) and to a lesser extent, (2) its estimated proportion of the total waste rock volume. The preliminary geochemical testing suggested that the large majority of the waste rock (79 percent) will be comprised of Group 3 materials with negligible potential for acid generation. Therefore, less emphasis was placed on correlating the number of samples to the waste or mine wall rock volume to avoid misdirecting the sampling program toward rock types which have negligible potential for acid generation. To provide a more conservative and potentially more effective waste characterization program, the focus of additional sample collection was directed toward identifying and characterizing the waste or mine wall rock lithologies with the highest potential for acid generation in an uncontrolled environment. Consequently, the number of samples collected for a given rock type does not correlate directly with its estimated proportion of the total volume.

Twenty clastic, three intrusive, three limestone and four low grade ore samples were submitted for acid base accounting analysis. Table 3-1 provides a summary of the samples submitted.

3.2 PHASE V - SECONDARY ACID BASE ACCOUNTING ANALYSIS

Clastic

Total sulfur values for the 20 clastic samples range from 2.02 to 8.29 percent with a geometric mean of 4.12, while the ANP values range from 9.6 to 109 $TCaCO_3/kT$ with a mean of approximately 60 $TCaCO_3/kT$. The resulting NNP values for the clastic samples range from -249.4 to 34.3 $TCaCO_3/kT$ with an average value of approximately -63 $TCaCO_3/kT$ suggesting that the clastic wall rock has a net acid generating potential.

TABLE 3-1

SUMMARY OF SAMPLES SUBMITTED FOR PHASE V ACID BASE ACCOUNTING ANALYSIS

Sample No.	Hole No.	Rock	Core Interval	North	East	Elev. (ft)	Sample Type	Stope Wall	Description
1	C-170	Clastic	335-360	26100	7220	2840	Pulp	East	Clastics, trace silica, 1% pyrite
2	C-138	Clastic	500-510	26100	7385	3010	Pulp	East	Clastics, trace silica, trace $CaCO_3$, 0.5 % pyrite, trace chalcopyrite
3	C-166	Clastic	181-186	26300	7367	3060	Pulp	East	Clastics, trace pyrite as blebs and tiny veinlets, trace silica
4	C-142	Clastic	475-480	26300	7335	2978	Pulp	East	Clasties, 2% pyrite with trace chalcopyrite in tiny quartz veinlets
5	C-147	Clastic	275-280	26500	7200	2922	Pulp	East	Clastics, 2% pyrite as veinlets, 1% magnetite as blebs, trace hematite, weak silicic
6	C-178	Clastic	118-123	26500	7325	3080	Pulp	East	Clastics, 3% pyrite quartz-calcite veinlets, trace chalcopyrite, weak silicic
7	C-100	Clastic	263-268	26700	7185	2995	Pulp	East	Clastics, 20% pyrite with 5% magnetite in quartz veinlets, weak silicic, very chloritic
8	C-066	Clastic	311.5-316.5	26700	7244	2845	Pulp	East	Clastics, 1% pyrite with 0.5% chalcopyrite in quartz veinlets, moderately chloritic
9	C-105	Clastic	186-191	26900	7163	2965	Pulp	East	Clastics, 3% pyrite as patchy clots, trace chalcopyrite with quartz veinlets
10	C-086	Clastic	255-260	26900	7209	2794	Pulp	East	Clastics, 2% pyrite with <1% chalcopyrite with quartz as veinlets, trace $CaCO_3$
11	C-085	Clastic	181.5-185	26900	7165	2910	Pulp	East	Clastics, 2% pyrite with trace chalcopyrite in quartz veinlets, very chloritic
12	C-102	Clastic	78-83	26900	7245	3050	Pulp	East	Clastics, 15% pyrite with trace chalcopyrite as veins, slightly chloritic

TABLE 3-1
(Continued)

Sample No.	Hole No.	Rock	Core Interval	North	East	Elev. (ft)	Sample Type	Stope Wall	Description
13	C-080	Clastic	90-95	27100	7135	2915	Pulp	East	Clastics, 3% pyrite with trace chalcopyrite in quartz veinlets
14	C-141	Clastic	345-350	27100	7160	2720	Pulp	East	Clastics, trace pyrite as tiny veinlets
15	C-184	Clastic	384-389	27100	7177	2618	Pulp	East	Clastics, trace pyrite as disseminations
16	C-095	Clastic	280-285	27300	7071	2550	Pulp	East	Clastics, <5% pyrite, in quartz veinlets
17	C-096	Clastic	359-365	27350	7017	2437	Pulp	East	Clastics, <5% pyrite in quartz veinlets, slightly silicic
18	C-116	Clastic	247-252	27300	7130	2660	Pulp	East	Clastics, 5% pyrite with 2% chalcopyrite in quartz veinlets
19	C-151	Clastic	194.5-200	27500	6981	2605	Pulp	East	Clastics, 10% pyrite with trace chalcopyrite in quartz veinlets
20	C-149	Clastic	255-260	27500	6922	2514	Pulp	East	Clastics, 2% pyrite with quartz in stockworks and disseminated, trace $CaCO_3$
21	C-191	Intrusive	363-370	26200	7148	2723	Core	West	Biotite rhyodacite, propyllitic altered, trace $CaCO_3$ as veinlets
22	C-140	Limestone	380-387.5	26200	7247	2940	Core	West	Limestone, $CaCO_3$ veining, bleached
23	C-178	Low Grade Ore	78-80	26500	7287	3087	Core	West	Altered limestone, 4% pyrite, 10% magnetitenetite, 1% pyrrhotite, 80% hematite, trace $CaCO_3$, slightly silicic
24	C-304	Intrusive	379-384	26400	7248	2790	Core	West	Hornblende, rhyodacite, trace pyrite disseminated
25	C-089	Intrusive	344-350	26700	7255	2768	Core	West	Hornblende, rhyodacite, no mineralization
26	C-066	Limestone	248-254	26700	7180	2855	Core	West	Limestone, 2% pyrite as wispy masses, 8% magnetite as patches, minor $CaCO_3$
27	C-110	Intrusive	324-328	26950	7171	2650	Core	West	Altered rhyodacite, argillic

TABLE 3-1
(Concluded)

Sample No.	Hole No.	Rock	Core Interval	North	East	Elev. (ft)	Sample Type	Stope Wall	Description
28	C-160	Low Grade Ore	353-358	26950	7126	2767	Core	West	Altered limestone, trace pyrite disseminations, 50% magnetite, 35% hematite, trace $CaCO_3$, silicic
29	C-127	Low Grade Ore	239.5-244	27150	7065	2743	Core	West	Altered limestone, trace pyrite, 2% magnetite, 40% hematite, weakly silicic
30	C-131	Limestone	262-266.5	27350	7038	2546	Core	West	Limestone, bleached, trace pyrite disseminated

Fourteen of the 20 samples have NNP values less than -20 $TCaCO_3/kT$ and are subsequently classified as Group 1 materials. Three samples are considered to have uncertain acid generation potential and are classified as Group 2 rocks, while two samples have a low to negligible potential for acid generation. Table 3-2 provides a summary of the acid base accounting results relative to the classification criteria. The ANP:AGP values are predominantly less than one and suggest that the majority of samples have acid generation potential.

The total sulfur content of the Phase V clastic samples appears to be higher than those samples submitted for Phase III testing. The mean total sulfur content of the initial clastic samples is 2.57 percent with a low value of 0.73 percent.

Limestone

Total sulfur values for the three limestone samples range from 0.13 to 0.51 percent and average 0.23 percent. ANP values range from 791 to 1020 $TCaCO_3/kT$ indicating that the limestone has vast neutralization capacity. Table 3-2 provides a summary of the acid base accounting results for the limestone samples. The most recent limestone samples are classified as Group 3 materials with NNP values an order of magnitude greater than 20 $TCaCO_3/kT$ and ANP:AGP values well in exeess of 3.0. The most recent data suggest that there is little variation in sulfide content and more importantly the neutralization capacity appears to be consistent within the unit as well.

Low Grade Ore

Three low grade ore samples were submitted for acid base accounting consisting primarily of altered limestone. Total sulfur values range from 0.21 to 4.17 percent with a geometric mean of approximately 0.57 percent. ANP values range from 429 to 639 $TCaCO_3/kT$ suggesting that the low grade ore has a large acid neutralization potential. Table 3-2 presents the acid base accounting results for the low grade ore samples. Based on the Phase V results, the low grade ore samples are classified as Group 3 materials with NNP and ANP:AGP values ranging from 299 to 633 $TCaCO_3/kT$ and 3.3 to 96, respectively. These results are consistent with the preliminary Phase III results for the low grade ore unit.

Intrusive

Total sulfur values of the four intrusive samples submitted for acid base accounting range from 0.06 to 1.2 percent with ANP values ranging from 207 to 308 $TCaCO_3/kT$. As a result, the intrusive samples are classified as Group 3 materials with a low to negligible acid generation potential and significant acid neutralization potential. The propylitic alteration of the intrusive unit is most likely the source of the large amount of carbonate mineralization present in these samples. These results agree well with those generated during the initial characterization.

Phase V Conclusions

The results of Phase V testing indicate that the limestone, low grade ore and intrusive rock types have a negligible potential for acid generation and are classified as Group 3 material. The clastic samples submitted indicate that this rock type has a potential to generate acid with 70 percent of the samples classified as Group 1 materials. The total sulfur values of the Phase V clastic samples were higher than those collected during Phase III, indicating that sulfide contents within this unit vary spatially relative to proximity to the mineralized zone. The acid base accounting data for the limestone, low grade ore and intrusive units appear to agree well with the initial results, suggesting little variability within these rock types.

TABLE 3-2

ACID BASE ACCOUNTING RESULTS FOR PHASE V TESTING

Sample #	Sample Name	Sample Type	Total Sulfur (%)	AGP (TCaCO3/kT)	ANP (TCaCO3/kT)	NNP (TCaCO3/kT)	ANP:AGP RATIO
1	91-LFC-170	Clastic	2.02	63.1	55.2	-7.9	0.9
2	91-LFC-138	Clastic	3.65	114.1	11.6	-102.5	0.1
3	91-LFC-166	Clastic	5.45	170.3	67.5	-102.8	0.4
4	91-LFC-142	Clastic	2.1	65.6	28.8	-36.8	0.4
5	91-LFC-147	Clastic	4.31	134.7	109	-25.7	0.8
6	91-LFC-178	Clastic	6.88	215.0	47.2	-167.8	0.2
7	91-LFC-100	Clastic	8.11	253.4	28.6	-224.8	0.1
9	91-LFC-105	Clastic	2.5	78.1	74.6	-3.5	1.0
10	X1EB0103.011	Clastic	2.77	86.6	336	249.4	3.9
11	X1EB0102.018	Clastic	7.29	227.8	168	-59.8	0.7
12	91-LFC-102	Clastic	8.4	262.5	141	-121.5	0.5
13	XOEB0101.335	Clastic	2.21	69.1	63.6	-5.5	0.9
14	91-LFC-141	Clastic	5.64	176.3	22.2	-154.1	0.1
15	91-LFC-184	Clastic	2.95	92.2	52.9	-39.3	0.6
16	91-LFC-95	Clastic	5.21	162.8	88.7	-74.1	0.5
17	91-LFC-96	Clastic	2.55	79.7	114	34.3	1.4
18	91-LFC-116	Clastic	8.29	259.1	9.6	-249.5	0.0
19	91-LFC-151	Clastic	6.95	217.2	86.5	-130.7	0.4

TABLE 3-2
(Concluded)

Sample #	Sample Name	Sample Type	Total Sulfur (%)	AGP (TCaCO3/kT)	ANP (TCaCO3/kT)	NNP (TCaCO3/kT)	ANP:AGP RATIO
20	91-LFC-149	Clastic	3.25	101.6	77.1	-24.5	0.8
26	XOEB0102.348	Clastic	2.57	80.3	69.6	-10.7	0.9
	ARITHMETIC MEAN		4.66	145.47	82.59		0.74
	GEOMETRIC MEAN		4.12	128.87	60.00		0.47
21	91-LFC-191	Intrusive	0.15	4.7	227	222.3	48.3
22	91-LFC-140	Limestone	0.20	6.2	1020	1014	164.5
23	91-LFC-178	Low Grade Ore	4.17	130	429	299	3.3
24	91-LFC-304	Intrusive	1.20	37.5	252	194.5	6.2
25	91-LFC-089	Intrusive	0.12	3.8	308	304.2	81.0
26	91-LFC-066	Limestone	0.51	15.9	791	775.1	49.7
27	91-LFC-110	Intrusive	0.06	1.9	207	205.1	108.9
28	91-LFC-160	Low Grade Ore	0.21	6.6	596	589.4	90.3
29	91-LFC-127	Low Grade Ore	0.21	6.6	639	96.8	96.8
30	91-LFC-131	Limestone	0.13	4.1	1020	1015.9	248.7

3.3 PHASE VI - IDENTIFICATION OF GEOCHEMICAL DOMAINS

Based on the Phase III acid base accounting results and information obtained from logging the core samples (Phase I), the lithological classification of the rock types was refined to reflect geochemical homogeneity and acid generation potential. As a result, four geochemical domains were identified;

- Limestone
- Clastic II
- Low Grade Ore
- Intrusive

These domains are believed to reflect, to the degree possible, the geochemical variability within the rock types present at Lamefoot which will comprise the waste material and wall rock. Based on the Phase V results the variability within each of the units is low. Hence, additional domains were not identified. The Clastic II designation was chosen for the clastic rock type primarily because of the degree of sulfide mineralization. This classification is consistent with the means of differentiating clastic domains at the Key Expansion.

3.4 PHASE VI CONCLUSIONS

Mine wall rock mineralogy and structural features control geochemical behavior relative to acid generation, acid neutralization, and leachability. The following physical rock features influence geochemical behavior:

- Nature, reactivity and surface area of sulfide grains (i.e., massive or disseminated);
- Fracture spacing and degree of mineralization;
- Particle size surface area exposure and/or exposure area as a function of frequency of water bearing fractures;
- Nature, reactivity and surface area of neutralizing minerals; and,
- The physical setting of the potentially reactive wall rock including availability of oxygen and water for reaction and transport.

The four Lamefoot Project waste rock domains have the following significant characteristics relative to acid generation potential:

- **Limestone**: This rock type is weakly to strongly fractured and generally contains 1 to 2 percent sulfide minerals located primarily along fractures and in veinlets. These structures may result in a preferential exposure of sulfide minerals as the waste rock is mined, however, the presence of significant amounts of calcium carbonate should neutralize any potential acid generation.

- **Low Grade Ore:** The low grade ore consists of altered limestone is comprised primarily of massive magnetite and/or sulfide with fragments of unaltered limestone incorporated into the matrix. The low grade ore is strongly fractured and subsequently healed with calcite. While preferential exposure of sulfides may occur the effective surface area of exposed carbonate minerals will also increase during mining. Significant amounts of pyrite and minor pyrrhotite are found within the low grade ore. Hematite and limonite are pervasive in zones of oxidation suggesting that weathering may have reduced the reactivity of the sulfide minerals. Cumulative sulfide, magnetite and oxides typically occur in amounts greater than 5 percent occasionally reaching amounts of 50 percent. The presence of significant amounts of carbonate mineralization within this unit suggest that while oxidation of sulfide minerals may occur the solutions will be highly buffered.

- **Clastic:** The Clastic rock type contains sulfide percentages ranging from less than 1 percent to greater than 5 percent. The primary mode of occurrence is either as disseminations or within microfractures and veinlets. While veinlets often represent zones of weakness which can be preferentially fractured during mining, the sulfides disseminated within the rock mass will tend to remain relatively unexposed. If the Clastic rock becomes reactive it will be along fracture faces and not within the rock mass itself.

- **Intrusive:** This rock type generally contains only trace amounts of pyrite (0-1 percent). Generally, the pyrite is disseminated throughout the rock matrix as very fine grained poorly formed crystals. Although disseminated pyrite has a high surface area, and hence stronger reactivity, the total mass of pyrite exposed any given intrusive rock fragment face is generally far less than the total 1 percent. The actual percentage of total sulfur content that will be exposed and potentially reactive is likely to be insignificant. Pervasive propylitic alteration has resulted in significant amounts of calcite mineralization which occurs throughout the unit. Consequently, the intrusive unit is likely to provide neutralization capacity to the system.

In addition to mineralogy and rock structural features, the hydrogeological setting and configuration of the ultimate workings influence geochemical behavior. Groundwater flow through the unsaturated and saturated zones is controlled by the frequency and degree of interconnection of the fracture system. These characteristics exert significant control on the geochemical regime. Pyrite oxidation occurs both on the mine wall and on the fracture planes within the wall. If reactive pyrite is exposed on submerged fracture surfaces, the rate of oxidation is significantly reduced and may become negligible. As exposed pyrite oxidizes in the unsaturated fractures, the resulting acidity will either be flushed away periodically or will be retained.

3.5 PHASE VII - LITHOLOGIC AND GEOCHEMICAL CORRELATION WITH KEY EXPANSION

The purpose of Phase VII of the Lamefoot waste characterization program is to identify potential lithologic and geochemical similarity between the waste rock domains present at Lamefoot and their Key Project equivalents.

The geologic/lithologic similarity between the two sites, and the usefulness of ABA data to properly predict anticipated geochemical behavior of the various waste rock domains provide the focus for this portion of the Lamefoot waste characterization program. Much of the focus of the Lamefoot program is centered on establishing both the geologic/geochemical similarities and variabilities between the waste rock domains present at both sites in order to facilitate utilization of existing data and interpretations to direct the Lamefoot Project geochemical program and potential mitigative design.

3.5.1 Lithologic Correlation

Information gathered during the preliminary geologic investigation and the metallurgical/mineralogical studies were used to confirm the lithologic and mineralogic similarity between the waste rock lithologies at the Lamefoot and Key Expansion locations. In addition, the similarity (or differences) in those factors such as alteration, structure, mineralization, and carbonate occurrence which influence geochemical behavior was also established. Tables 3-3 and 3-4 present the lithologic descriptions and secondary features for the waste rock present at both the Lamefoot and Key Expansion Projects.

TABLE 3-3

SUMMARY OF WASTE ROCK LITHOLOGIES PRESENT AT THE KEY EXPANSION AND LAMEFOOT PROJECTS

Lithologic Unit	Geologic Description		Lithologic Similarities	Lithologic Differences
	Location			
	Key Project	Lamefoot Project		
Limestone	Dark gray to white, coarse grained to aphanitic, primarily composed of calcite with minor amounts of dolomite, massive with zones of extensive micro-fracturing. Dark gray portions commonly contain local zones of interbedded light gray to white limestone. Commonly sparitic with numerous styolites occurring throughout. Pyrite and pyhrrotite comprise the sulfide minerals and occur locally in fractures ranging from 0 to 1 percent. Magnetite occurs locally in fractures and disseminated within the matrix. The texture of the magnetite ranges from massive clots to fine grained. Magnetite abundance ranges from 0 to 15 percent.	Dark gray to light gray, with occasional interbedded layers of light colored limestone, coarse grained to aphanitic in texture, massive with zones of local fracturing. Abundant calcic styolites and veining. Pyrite and chalcopyrite with minor amounts of pyhrrotite comprise the sulfide minerals and occur locally in fractures and veinlets. Sulfide percentages range from 0 to 1 percent. Magnetite occurs within veins and veinlets and where mineralization is not limited to structure, the magnetite is occasionally disseminated throughout the matrix. Magnetite abundance ranges from 0 to 15 percent.	Primarily composed of calcite, interbedded zones of light and dark limestone, commonly sparitic with abundant styolites. Sulfides occur in fractures and veinlets in trace quantities, magnetite occurs in fractures, veinlets and disseminated in the matrix.	None

TABLE 3-3
(Continued)

Lithologic Unit	Geologic Description — Location — Key Project	Geologic Description — Location — Lamefoot Project	Lithologic Similarities	Lithologic Differences
Low Grade Ore	Bronze to black massive sulfide and/or magnetite. Sulfides consist of pyrite and phyrrotite. Low grade ore commonly is found as sulfide and magnetite horizons grading between one another. Textures range from fine grained to coarse grained. Sulfides occur in the magnetite as fine to medium clots, within veinlets and disseminated throughout the matrix. Magnetite occurrence in the massive sulfides is limited to fractures and crude bands. The low grade ore matrix is generally calcareous, weakly to strongly preserved. Hematite and limonite are common in the zones of oxidation.	Massive sulfide and/or magnetite, the magnetite consists of swirled bands of magnetite, hematite, and calcite with minor pyrite. Typically the magnetite contains fragments of unaltered limestone. Sulfides consist of massive to fine grained pyrite with variable amounts of magnetite. Pyrite occurrence in the massive magnetite is typically within stringers, veinlets, or disseminated throughout the matrix. Alteration of the low grade ore consists of chlorite and hematite with variable amounts of limonite. Calcite occurs within veins and disseminated through out the matrix.	Both low grade ores contain massive sulfides and magnetite. The matrix of the low grade ore is calcareous and weakly to strongly preserved. Hematite and limonite are common in oxidation zones.	Lamefoot contains fragments of unaltered limestone. Key massive sulfides contain more phyrrotite.
Intrusive	Pale green to gray, aphanitic to coarse grained, locally porphyritic, contains horneblende, biotite and variable amounts of plagioclase and potassium feldspar. Alteration consists of weak to strong chloritization. Fracturing ranges from moderate to strong and often calcite heals fractures. Sulfide abundance ranges from 0 to trace amounts (< 1%) and primarily occurs disseminated throughout the groundmass.	Pale olive to dark gray, aphanitic vitreous groundmass, containing biotite phenochrysts. Weak to strong propylitic and/or chloritic alteration, moderate to strong fracturing infilled with gouge or healed with calcite. Trace (<1%) amounts of pyrite typically occur disseminated throughout the groundmass.	Similar textures and composition. Both exhibit chloritic alteration. Similar structure with calcite healing fractures. Sulfide abundance is trace and occurs disseminated.	Lamefoot contains biotite phenocrysts. Alteration of Lamefoot includes propyllitic.

TABLE 3-3
(Concluded)

Lithologic Unit	Geologic Description Location Key Project	 Lamefoot Project	Lithologic Similarities	Lithologic Differences
Clastic	Dark green to tan interbedded sharpstone conglomerate, siltstone, mudstone and cataclastite. Siltstones and mudstones are often interbedded as thin laminae. The siltstones exhibit well developed micro-fractures and shears and contain angular to subangular chert clasts. The conglomerate ranges from fine to coarse grained, containing subrounded to angular chert clasts supported in a siltstone matrix. Chloritic, silicic, and argillitic alteration ranges from weak to strong. In areas of pervasive alteration, bleaching of the unit is common. Pyrite occurs in micro-fractures and disseminated within quartz veins which are surrounded by bleached margins. Visual estimates of sulfide percentages range from trace to 2 percent, depending on the proximity to mineralization.	Light green to dark gray interbedded sharpstone conglomerate, siltstone, mudstone, and cataclastite. Moderate to strong fracturing occurs locally. Siltstone and mudstone have been altered locally to argillites with abundant quartz-pyrite veins. Pervasive alteration is chloritic with local strong hematitic alteration around fractures. Calcite often heals micro-fractures and shears. Bleached zones occur near areas of intense mineralization. Sulfides primarily occur within micro-fractures and disseminated throughout the matrix in zones of mineralization. Pyrite comprises the bulk of the sulfide mineralization and is generally fine grained to massive in texture. Sulfide abundance ranges from 0 to less than 10 percent.	Interbedded sharpstone conglomerate, siltstone, mudstone, and cataclastite. Pyrite composes the bulk of sulfide mineralization which occurs in micro-fractured quartz veins and disseminated throughout the matrix. Sulfide abundance is variable and is a function of proximity to the zone of mineralization.	Bleaching in areas of mineralization more common and distinct at Key. Alteration at Lamefoot is primarily chloritic while that at Key includes chloritic, silicic, and argillitic. Economic mineralization is significant at Lamefoot in areas adjacent to primary orebody, and these portions will be mined as ore.

TABLE 3-4

SUMMARY OF FACTORS INFLUENCING POTENTIAL GEOCHEMICAL BEHAVIOR OF WASTE/MINE WALL ROCK AT BOTH THE KEY EXPANSION AND LAMEFOOT PROJECTS

	Key Project				Lamefoot Project			
Lithologic Unit	Alteration	Structure	Sulfide Mineralization	Carbonate Occurrence	Alteration	Structure	Sulfide Mineralization	Carbonate Occurrence
Limestone	None or slightly dolomitized	Massive with moderate to strong fracturing and micro-fracturing.	Trace pyrite in micro-fractures and veinlets, occasionally disseminated.	Predominantly calcium carbonate with minor dolomitization.	None or slightly dolomitized	Massive with moderate to strong fracturing and micro-fracturing.	Trace pyrite in micro-fractures and veinlets, occasionally disseminated.	Predominantly calcium carbonate with minor dolomitization.
Low Grade Ore	Moderate to strong chlorite and calcite alteration.	Massive to broken, weakly to strongly fractured and micro-fractured.	Predominantly pyrite and phyrrotite occurring with massive and fine to coarse grained texture. Habit ranges from amorphous to euhedral depending on texture.	Calcite often occurs in the matrix especially in the massive magnetite. Also occurs in fractures and veins.	Calcic, hematitic, and limonitic alteration in zones of oxidation.	Massive to sheared, fractured and brecciated.	Predominantly pyrite and phyrrotite occurring with massive and fine to coarse grained texture. Habit ranges from amorphous to euhedral depending on texture.	Calcite often occurs in the matrix especially in the massive magnetite. Also occurs in fractures and veins.

TABLE 3-4
(Concluded)

	Key Project				Lamefoot Project			
Lithologic Unit	Alteration	Structure	Sulfide Mineralization	Carbonate Occurrence	Alteration	Structure	Sulfide Mineralization	Carbonate Occurrence
Intrusive	Predominantly chloritic alteration.	Weakly to strongly sheared and fractured. Local zones of rubble in brecciated areas.	Trace amounts of pyrite disseminated in groundmass. Occurring as very fine to fine grained amorphous to sub-euhedral crystals.	Calcite occurs in veinlets typically associated with alteration.	Pervasive propyllitic and chloritic alteration.	Weakly to strongly sheared and fractured. Local zones of rubble in brecciated areas.	Trace amounts of pyrite disseminated in groundmass. Occurring as very fine to fine grained amorphous to sub-euhedral crystals.	Calcite occurs in veins typically associated with propyllitic alteration.
Clastic	Argillitic and chloritic alteration.	Weakly to strongly sheared and fractured. Local zones of rubble in brecciated areas. Calcite commonly heals fractures.	Weakly to strongly disseminated pyrite. Very fine to coarse grained amorphous to euhedral crystals. Sulfides occur in fractures, micro-fractures and in veinlets.	Calcite occurs disseminated throughout the matrix and within veinlets and micro-fractures.	Argillitic and chloritic alteration.	Weakly to strongly fractured and sheared. Local areas of rubble where brecciated. Calcite commonly heals fractures.	Weakly to strongly disseminated pyrite. Very fine to coarse grained amorphous to euhedral crystals. Sulfides occur in fractures, micro-fractures and in veinlets.	Calcite occurs disseminated throughout the matrix and within veinlets and micro-fractures.

The major lithologic characteristics between the rock types present at the Lamefoot and Key Expansion sites correlate very well with one another. The paragraphs which follow summarize the lithologic similarities between the units at both sites.

Limestone

The limestone at both Lamefoot and the Key Expansion is predominantly calcium carbonate with minor amounts of dolomite. Both contain interbedded layers of light colored limestone and abundant calcareous styolites and veining. Magnetite occurs in fractures, veinlets, and as disseminations throughout the matrix at both Lamefoot and the Key Expansion. Pyrite is the major sulfide mineral with minor pyrrhotite. Sulfide abundance in the limestone is similar at both sites with the mode of occurrence primarily limited to micro-fractures and veinlets.

Low Grade Ore

The low grade ore present at both sites consists of massive sulfides and magnetite. Pyrite, along with minor pyrrhotite and chalcopyrite comprise the sulfides present in the low grade ore at both Lamefoot and the Key Expansion sites. The magnetite consists of swirled bands of magnetite, hematite, and calcite. At both sites hematite and limonite are commonly found in the zones of oxidation. The habit and occurrence of the sulfides present in both the Key and Lamefoot low grade ore are similar as is the degree of carbonate mineralization. The matrix of the low grade ore at both sites is calcareous and preserved to varying degrees.

Clastic

The clastic rocks present at both sites are comprised of interbedded sharpstone conglomerates, siltstones, mudstones, and cataclastites. Strong to moderate fracturing and micro-fracturing are common within this unit at both Lamefoot and the Key Expansion. Pyrite composes the bulk of the sulfide mineralization which occurs in micro-fractures, quartz veins, and disseminated throughout the matrix. Sulfide abundance in the clastic units at both sites is spatially variable and is generally a function of proximity to the zone of mineralization. Clastic rocks at both sites exhibit two geochemical domains; Clastic I and Clastic II.

There are two primary differences between the clastic rocks which are present at the Lamefoot site and those at the Key Expansion site. The clastic rocks at the Key Expansion site which are located adjacent to the mineralized zones characteristically show distinct bleaching and carry no significant economic mineralization. The clastic rocks at the Lamefoot site which are similarly located exhibit variable degrees of bleaching and contain significant amounts of economic mineralization. The Clastic II rocks at Lamefoot typically contain higher sulfide percentages and increased carbonate mineralization. No Clastic I rocks will be exposed in the ultimate workings at Lamefoot and as such it is not considered as a geochemical domain within the mine. However, ABA data for the backfill samples suggest that these materials are classified as Clastic I.

Intrusive

The Tertiary intrusive is consistent between the two sites in texture, composition, sulfide abundance and mode of sulfide occurrence. The texture of the intrusive rocks present at Lamefoot and the Key Expansion ranges from aphanitic to coarse and is composed of hornblende, biotite and variable amounts of plagioclase and potassium feldspar. Sulfide abundance ranges from 0 to less than 1 percent and is generally disseminated throughout the groundmass or found within fractures and microfractures along with calcite.

Summary of Lithologic Correlation

Minor differences exist between rock types in secondary features such as alteration and structure, however, the differences do not effectively limit the ability to correlate units. The geologic environments of both the Lamefoot and Key Expansion ore deposits are virtually identical therefore, the waste and mine wall rock domains are also correlative. This geologic similarity allows the full range of geochemical data generated for the Key Expansion to augment the current Lamefoot waste characterization program. One of the objectives of the technical approach adopted for the Lamefoot waste rock characterization program focused establishing the degree of variability (both geochemical and geologic) between the Lamefoot and Key Project waste rock domains.

Extensive work was performed as part of the Key Expansion waste characterization program to ensure that the geochemical variability of the various waste rock domains was adequately defined. This process included both acid base accounting and kinetic testing. Preliminary predictions regarding potential acid generation were developed for each of the waste rock domains by evaluating their respective acid base accounts relative to criteria involving the acid neutralizing potential and the acid generation potential (see Appendix E, Golder 1992).

The validity of using ABA data to predict the acid generation potential of the various waste rock domains was subsequently confirmed through the kinetic testing completed as part of the Key Expansion waste characterization program. The results of the kinetic testing program illustrated that the criteria established to define acid generation potential, based on ABA data for the various waste rock domains, were representative of actual geochemical behavior and appropriately conservative.

3.5.2 Geochemical Correlation

In order to define the degree of geochemical similarity between the Key Expansion and Lamefoot sites the acid base accounting data were evaluated relative to each of the units. In addition, both the kinetic and leachability data from the Key Expansion waste characterization program are discussed relative to the acid generation and metal leachability potential of each of the units present at the Lamefoot site.

3.5.2.1 Acid Base Accounting

Limestone

The average total sulfur content of the Key Expansion limestone is 0.05 percent while the average for the limestone present at the Lamefoot site is approximately 0.4 pereent. The acid neutralizing potential of the limestone at the two sites is similar with values of 871.3 and 786.1 $TCaCO_3/kT$ for Key and Lamefoot, respectively. As a result of these high neutralization potentials the NNP values for both sites are two orders of magnitude greater than 3, which is the criterion for negligible acid generation. Therefore, it can be concluded that these units are similar in their negligible potential for acid generation and provide the potential for neutralizing capability.

Low Grade Ore

The average total sulfur values for the low grade ore at the Key Expansion and Lamefoot sites are 4.81 and 0.13 percent, respectively. The acid neutralizing potential for the Key site is slightly lower with a value of 139 $TCaCO_3/kT$ as compared to 371 $TCaCO_3/kT$ for the low grade ore at Lamefoot. This suggests that the relative acid generation potential of the Key low grade ore is higher than present at the Lamefoot site. The difference in the low grade ore between the two sites is primarily in the degree of replacement in the limestone and the amount of unaltered parent material present. The low grade ore present at the Lamefoot site includes significant amounts of unaltered limestone clasts which increase its effective neutralization potential. Thus, it appears the low grade ore present at the Lamefoot site has a much lower potential for acid generation relative to its Key Expansion equivalent.

Clastic II

The Clastic II rock type has average total sulfur values of 1.02 and 3.81 percent at the Key Expansion and Lamefoot sites, respectively. The acid neutralizing potential of the Clastic II rocks at the Key site is 33.25 $TCaCO_3/kT$ as compared to 60.5 $TCaCO_3/kT$ at the Lamefoot site. Average net neutralization values suggest that the Key Clastic II rocks have an uncertain acid generation potential (-1.32 $TCaCO_3/kT$) while those at the Lamefoot site have a strong potential for acid generation (-58.5 $TCaCO_3/kT$). Comparison of the acid base accounting data for the Clastic II rocks at both sites suggest that the degree of mineralization within the Clastic II rocks found at the Lamefoot site is greater than that at the Key Expansion site. In addition, because of the mining method (i.e., open pit), the economic mineralization is much higher within the portions of the Clastic II rock at the Key site with similar sulfide contents to that of Lamefoot. Much of this portion of the unit was mined as ore, at the Key site and was not a significant portion of the waste rock volume. At Lamefoot the Clastic II rock will be left intact as wall rock in stopes.

Intrusive

Average total sulfur values for the intrusive rock at the Key and Lamefoot sites are 0.01 and 0.64, respectively. The intrusive rocks at both sites have significant acid neutralization potential ranging from 66.8 (Key) to 242 $TCaCO_3/kT$ (Lamefoot). The low total sulfur contents and the relatively high carbonate mineralization are common in the intrusive rocks present at both sites and is reflected in the high NNP values observed. Average NNP values for the Key and Lamefoot intrusive rocks are 171.6 and 220.0 $TCaCO_3/kT$, respectively. It is anticipated that the intrusive rocks present at the Key and Lamefoot sites are similar in their negligible potential for acid generation.

3.5.2.2 Kinetic Testing and EPA 1312 Leachability Testing

As part of the Key Expansion waste characterization program kinetic testing was completed on two Clastic I samples, two Clastic II samples, and one Low Grade Ore sample. The kinetic testing program and results are described in detail in both Golder 1992, and Golder 1993. The following sections discuss the results for the Key samples submitted for kinetic testing relative to their Lamefoot equivalents.

Kinetic Testing

Clastic I

Two Clastic I samples were submitted for kinetic testing and were run in humidity cells for a 20 week period. Initial pH values ranged from 7.44 to 7.55, while week 20 pH values ranged from 6.6 to 6.66. Alkalinity values remained stable throughout the 20 week testing period and ranged from 19 to 29 mg/l, suggesting that significant alkalinity depletion did not occur during this time. Sulfate production was low with concentrations below the detection limit (<10 mg/L) after the 20 week period. Cumulative sulfate production stabilized in both samples prior to the completion of the 20 week period indicating that oxidation of sulfide minerals ceased. In both samples cumulative alkalinity continued to increase at a rate greater than the sulfate production suggesting that the relative rates of reaction favored the dissolution of carbonate minerals. These samples were determined to be non-acid producing and it is anticipated that the behavior of the unmineralized Clastic I rocks at Lamefoot will be the same.

Clastic II

Two samples from the Clastic II domain were submitted for kinetic testing. The test duration differed for the two samples because uncertain geochemical behavior required that one sample be run for an additional 10 weeks to confirm the observed trends. Initial pH values ranged from 7.55 to 7.83 while final pH values ranged from 7.34 to 7.51. The relative stability of the pH values indicated that both samples were well buffered. Initial alkalinity values ranged from 28 to 36 mg/l and decreased to a common value of 19 mg/l at the end of testing. Cumulative alkalinity continued to increase through time suggesting that significant depletion of neutralization potential did not occur in either sample. Sulfate concentrations were low in both samples and at the end of testing were below the limits of detection. Cumulative sulfate production stabilized in both samples indicating that significant pyrite oxidation was not occurring. As a result, both samples were determined to be non-acid generating. The total sulfur values of these samples represent the low range of pyrite content for the Lamefoot Clastic II domain. As a result, it can be expected that while a portion of the Clastic II domain at Lamefoot may be classified as having a potential to generate acid based on ABA data its actual behavior will be non-acid generating.

Low Grade Ore

One Low Grade Ore sample was submitted for kinetic testing and was run for a 30 week period. The initial and final pH values for the Low Grade Ore were 7.38 and 7.18, respectively. Although there was some variation, no downward trends in pH were evident suggesting that the sample was well buffered. The initial alkalinity after one week was 24 mg/l. Subsequent to 30 weeks of testing the alkalinity decreased to 16 mg/l. Cumulative alkalinity continued to increase through time indicating that carbonate mineralization remained available for dissolution after 30 weeks of testing. Sulfate concentrations remained low throughout the duration of testing and after 30 were below detectable limits. Cumulative sulfate increased throughout the testing period until the final week when sulfate concentrations fell below detection. This suggests that pyrite oxidation was occurring but that the sample remained buffered because of the available neutralization capacity. This sample was designated as non-acid generating. Because this sample had a sulfur content higher than the average Lamefoot Low Grade Ore and a lower neutralization potential it is anticipated that the Lamefoot Low Grade Ore will exhibit the same non-acid generating behavior.

EPA 1312 Leachability Testing

The leachability characteristics of the five samples submitted for kinetic testing were defined using EPA Method 1312. The objective of this testing was to evaluate and classify the potential for significant metals leaching under acidic conditions. The complete discussion of results appears in Appendix E of Golder (1992).

All five of the samples tested produced leachates with moderately basic pH values which ranged from 8.26 to 8.92. Conductivity values ranged from 59 μmhos/cm to 77 μmhos/cm. Of the 26 metal species analyzed, 7 were above the detection limit in at least one sample. These species include aluminum (4 samples), barium (2 samples), calcium (5 samples), magnesium (5 samples), molybdenum (1 sample), strontium (5 samples), and zinc (1 sample). All species were present in trace amount with the exception of aluminum, calcium, and magnesium. Sulfate and iron were below detection limits in all five samples. The moderate pH values and low metals concentrations suggest that the Clastic and Low Grade Ore domains have low leachability characteristics. It is anticipated that given the general similarity in overall mineralogy of these units at both the Key and Lamefoot sites that the potential for significant leachability from the underground workings at Lamefoot is low.

3.5.2.3 Phase VII - Conclusions

The purpose of Phase VII of the Lamefoot waste characterization program is to identify potential lithologic and geochemical similarity between the waste rock domains present at Lamefoot and their Key Expansion equivalents.

A comparison of the acid base accounting data for both the Key Expansion and the Lamefoot sites suggest that while there is some variability, the Limestone and Intrusive domains reflect distinct similarities and as a result share a negligible potential for acid generation. While the Low Grade Ore domains differ between sites, the Lamefoot Low Grade Ore unit appears to have a low to negligible potential for acid generation. This is based on a comparison of both static and kinetic data. The Low Grade Ore sample submitted for kinetic testing from the Key site has a high potential to generate acid based on the static test data. This potential was not confirmed through the kinetic test data which indicated a well buffered system with low sulfate production and neutral pH values. Because the static test data for the Lamefoot low grade ore indicate a low potential for acid generation it is not anticipated that this rock type will be of environmental concern even in an uncontrolled setting.

The results of the Key Expansion kinetic testing program suggest that both the Clastic I and Low Grade Ore domains present at Lamefoot will be non-acid generating. The Clastic II domain appears to differ to a significant degree between the two sites indicating that a direct correlation may be difficult. Although the Clastic II samples tested from Key did not produce acid, extrapolating the results to the Lamefoot Clastic II unit may be inappropriate because of the differences in the degree of mineralization. As discussed in Section 3.5.2.1, total sulfur percentages for the Lamefoot Clastic II domain were slightly higher than those for the Key Expansion. As a result, the average NNP values for the Lamefoot Clastic II samples suggest that a strong potential exists for acid generation if the conditions in the mine are favorable for reaction and transport.

The following section discusses the controlling factors which influence both acid generation and contaminant transport in a fractured underground mine setting. In addition, the technical approach to evaluating the need for, and applicability of, various mitigative alternatives is also presented.

4.0 ACID DRAINAGE FROM MINE WALLS AND MITIGATIVE APPROACH

4.1 ACID DRAINAGE FROM MINE WALLS

A key factor in understanding the potential for acid drainage from mine walls is the knowledge of water movement in and near the mine walls. Sulfide oxidation only becomes an environmental concern when products such as hydronium ions, sulfate and metals are transported away from the reaction sites and into the local environment.

Two basic sources of water near the mine wall are groundwater and surface water, which may be interactive depending on the local hydrogeologic regime. Because of the limited connection between the ground surface and the deeper fractured bedrock aquifer, surface water is considered a negligible consideration at the Lamefoot site. In the site vicinity, deep groundwater migrates in fractures which are either saturated with water or partially unsaturated. The geochemical implications of partial or complete saturation are significant relative to potential environmental impacts. Some case studies have indicated that the reactivity of sulfide minerals becomes negligible under saturated conditions while a significant quantity of acidity can be stored in unsaturated portions of the mine walls (Morin 1990). This stored acidity can be available for transport to the receiving environment as a result of intermittent flushing or resaturation of the system.

Acid drainage and accelerated metal leaching are consequences of oxidation of sulfide minerals which results in an acidic environment around the sulfur-bearing metals. Water movement over the oxidation site mobilizes the acidity, sulfate and leached metals. The physical characteristics which constrain the movement of water in the system will ultimately control the potential for environmental impacts from sulfide oxidation. Therefore, the conceptual hydrogeologic model developed for the Lamefoot site (Section 1.3) has a primary role in evaluating the need and the potential effectiveness of various mitigative alternatives.

The following characteristics of the Lamefoot mine hydrogeology influence the potential for environmental impacts related to AMD:

- Frequency and degree of interconnection between water bearing fractures and joints;
- Rates and magnitude of groundwater inflow to the mine;
- Post-mining hydraulic equilibrium conditions and the resulting potential for mine outflow; and finally,
- Hydrologic connection of potentially acid generating unit(s) relative to the flow regime.

The following paragraphs discuss the Lamefoot mine hydrology relative to these characteristics.

The fracture and jointing characteristics controlling groundwater inflow into the mine relate primarily to igneous intrusions and the associated fractures in the country rocks. Nearly all the observed mine inflow originates from the contacts between the igneous intrusions and the surrounding limestone. Intrusive rocks are distributed throughout the limestone unit west of, and within the Anfo Fault. The occurrence of the intrusive rocks is structurally controlled by flat lying reverse thrust faults and steeply dipping normal faults. Localized intrusions typically result in fracturing of the adjacent country rock because of thermal and mechanical forces. Intrusion of the igneous bodies most likely enhanced the local permeability of these areas by increasing the degree of connection between existing fractures in the limestone. In addition, the relatively low permeability intrusions act as flow barriers which enhances the lateral migration of groundwater to the local, highly fractured margins adjacent to the intrusion. As a result, in areas within the Low Grade Ore for example, where flow has occurred in the recent past, iron staining is common. This staining reflects preferential flow patterns within the fracture/joint system and may indicate areas of unsaturation which are subject to intermittent flushing.

While the Clastic rocks contain intrusive bodies, they occur with much less frequency. Most significantly however is the lack of hydraulic control they exert on the occurrence and migration of water within this unit. Currently, no groundwater inflow has been observed either from the intrusive/clastic contacts or from fractures or joints within the Clastic. No iron staining has been observed on any of the Clastic exposures indicating that at no time in the recent past has water movement occurred along the walls.

Recent development drifting east of the Anfo Fault indicates that nearly all faults and joint sets observed in the Clastic rocks are healed with quartz mineralization or fine sericitic clay minerals. These healed faults effectively act as barriers to groundwater flow. In addition, the geometry and number of intersections of joints and faults present in the Clastic unit is significantly different than in the limestone, suggesting that a well developed, open fracture/joint system is not present within the Clastic rock greatly limiting the potential for transmitting groundwater. The primary permeability of the clastic rocks is extremely low because of low grade metamorphism which resulted in recrystallization of mineral grains and indicates that fracture flow rather than matrix flow will predominate, if at all.

The topographic location of the clastic rocks also influences its hydrostratigraphic significance. The clastic unit outcrops at a topographic high above the mine workings and as a result may be isolated from notable recharge. Effects of topography relative to the movement and occurrence of groundwater are typically related to the greater number and larger aperture size of fractures and more effective weathering in ravines than on upland areas. In contrast, the limestone unit outcrops along the side of the ridge and along the valley floor near Wolf Camp Road suggesting that recharge to this unit is potentially much more significant than to the Clastic unit. Limited recharge to this unit further reduces its potential for environmentally significant acid mine drainage.

These hydrogeologic characteristics influence the potential for AMD in the following ways:

- Flow into the mine is controlled entirely by the Limestone unit and Low Grade Ore indicating that the groundwater has significant amounts of alkalinity and buffering capacity;

- Flow into the mine from the Clastic rocks does not occur indicating that one of two primary agents for sulfide oxidation (i.e., oxygen and water) is not present and transport away from the reaction site is unlikely even if limited oxidation does occur;

- While the Clastic rock type has a potential for acid generation under the proper conditions, the lack of primary permeability and negligible secondary permeability (i.e., open fractures) effectively reduce the risk of acid generation from this unit to very low levels. Even if fracture flow was observed in the Clastic rock, the actual mass of reactive rock contributing to acid generation would be limited to the reaction halo adjacent to the fracture and as a result would be much lower than the total mass of potentially acid generating wall rock;

- Because of limited recharge to the Clastic rock, healed fractures and extremely low matrix porosity, the opportunity for potentially reactive sulfide minerals to come into contact with appreciable water is very low. Hence, transport of environmentally significant concentrations of potential contamination is not likely to occur.

4.2 DEVELOPMENT OF THE MITIGATIVE APPROACH

The mitigative approach currently being developed for the Lamefoot mine has two components; prevention and abatement methods. Prevention methods include designs incorporated into the mine plan which are developed with the acid generation potential of the wall rock in mind. Abatement methods include measures which can be implemented if the prevention design is not sufficiently effective. The following sections provide a discussion of each of these components.

4.2.1 Prevention of AMD

The control of AMD by preventing or inhibiting acid generation is the most efficient and environmentally sound level of control. The objective of acid generation control is to prevent or reduce the rate of acid formation at the source. The development of the Lamefoot mine will preserve to the degree possible, the natural hydrogeologic conditions which are favorable to preventing sulfide oxidation. As mentioned above, the Clastic unit possesses the greatest potential for acid generation. However this unit is also the least significant in terms of the hydrogeologic regime to the extent that groundwater is not transmitted through this unit into the mine.

Groundwater seepage into the mine will be minimized. By utilizing a mine design that reduces the potential for long-term subsidence and loss of hydraulic integrity within the Clastic unit. Echo Bay is committed to utilizing construction and mining methods which will result in conditions which will reduce or prevent AMD (Plan of Operations, Echo Bay 1993).

Additional site characteristics which will facilitate the prevention of AMD include highly alkaline groundwater which will inhibit acid generation. Because the groundwater seepage into the mine occurs entirely within the limestone and low grade ore the alkalinity values are on the order of 300 to 400 mg/l as

$CaCO_3$, with an average of approximately 330 mg/l as $CaCO_3$. This represents approximately 0.77 pounds of $CaCO_3$ for every 1000 liters of groundwater. Therefore significant insitu neutralizing capacity exists in the system which will contribute additional protection against the formation and migration of AMD.

In order to prevent disturbing potentially acid generating material in the area of the backfill excavation, additional drilling and sampling will be completed to define the areal extent of this rock. The excavation will be completed such that this material will be left in place to ensure to the degree possible that ARD formation does not occur.

4.2.2 Abatement of Acid Mine Drainage

As part of the mine design additional pH control will be affected by backfilling the mine workings with approximately 230,000 tons of limestone and low grade ore (i.e., altered limestone). The addition of excess alkaline material will provide further control in preventing the formation and/or migration of potential AMD into the receiving environment. If it is assumed that the Calcium Carbonate Effectiveness (CCE) of the limestone and low grade ore is on the order of 70 percent then the actual alkalinity available for pH control is approximately 161,000 tons $CaCO_3$. This represents approximately 1,400 pounds $CaCO_3$/ton limestone.

The combined reaction of acid generation from pyrite by air/water oxidation and subsequent reaction with calcite is described by the following equation:

$$4\ FeS2 + 8\ CaCO_3 + 15\ O_2 + 6\ H_2O \rightarrow 4\ Fe(OH)_3 + 8\ Ca^{+2} + 8\ CO_2$$

Effectively, it requires two moles of calcite to neutralize the acid generated by a single mole of pyrite. This is equivalent to 200 grams of calcite for 120 grams of pyrite on a mass basis. Therefore, the addition of 161,000 tons of $CaCO_3$ will neutralize the acidity generated by the oxidation of approximately 97,000 tons of pyrite. If it is assumed that the average pyrite content of the exposed Clastic unit is 4.12 percent then the amount of reactive rock required to generate enough acid to overwhelm the potential neutralizing capacity of the limestone backfill is approximately 2.3×10^6 tons.

Assuming an exposure area of approximately 466,700 ft^2 for the Clastic II domain in the ultimate mine workings and a bulk density of 140 lbs/ft^3, the thickness of the reactive halo around the exposed rock would be approximately 71 feet. Based on values cited in the literature, typical thicknesses for reaction halos along fracture surfaces are on the order of 20 cm (0.65 ft) (Morin 1990). This is a highly variable parameter however, which is controlled by the rate of water movement, the efficiency of product removal through flushing, potentially most importantly the primary permeability of the matrix material. If the rock is highly indurated such as the Clastic lithology, water and air will be limited to reacting with oxidation sites along the fracture face and not within the matrix of the rock (i.e., internal reaction). Because potential acid generation from the Clastic domain is most likely dominated by reactions on fracture surfaces the amount of reactive pyrite exposed to air and water is much less than is required to overwhelm the neutralization capacity of the limestone backfill.

The availability of excess neutralizing capacity combined with the apparent low percentage of reaction sites within the Clastic wall rock suggest that the net acid generation potential of the system is low under current conditions.

In summary, the objectives of the mitigative approach developed for the Lamefoot mine are as follows:

- Utilize the characteristics of the existing hydrogeologic regime to prevent or minimize acid generation;

- Minimize disturbance of the existing hydrogeologic conditions, especially relative to the Clastic hydrostratigraphic unit, by implementing construction and mining practices designed to preserve, to the degree possible, the existing conditions which are favorable to preventing acid mine drainage;

- Utilize limestone backfill to achieve additional acid generation control at the potential source in order to retard the rate of acid generation to an environmentally insignificant level;

- Utilize additional acid neutralizing backfill from surface;

- Implement selective excavation using visual and qualitative methods of estimating acid generation potential of the backfill during an ongoing monitoring program; and,

- Plug all openings to the mine workings with air/water tight cement seals.

5.0 REFERENCES

British Columbia Acid Mine Drainage Task Force. 1989. Draft Acid Rock Drainage Technical Guide, Volume 1.

Brodie, M.J., Broughton, L.M, and MacG, Dr. A. 1991. A Conceptual Rock Classification System for Waste Management and a Laboratory Methods for ARD Prediction from Rock Piles. Second International Conference on the Abatement of Acid Drainage, Montreal, Quebec, Canada.

Echo Bay Minerals. 1993. Lamefoot Plan of Operations, April 1993 Modification with Supplemental Information. July.

Farmer, Gene. 1992. A Conceptual Waste Rock Sampling Program for Mines Operating in Metallic Sulfide Bores with a Potential Acid Rock Drainage. Draft. U.S.D.A. Forest Service.

Golder. 1992. Final Environmental Impact Statement, Kettle River Key Project Expansion.

Golder. 1993.

Morin, K.A. 1990. Acid Drainage from Mine Walls the Main Zone Pit at Equity Silver Mines. British Columbia Acid Mine Drainage Task Force.

Sobek, A.A., W.A. Schuller, J.R. Freeman, and R.M. Smith. 1978. Field and Laboratory Methods Applicable to Overburden and Minesoils. United States Environmental Protection Agency, EPA 600/2-78-054.

Smith, A. 1992. Personal Communication. Telephone Conversation with Adrian Smith of ASCI, Inc. and John Chahbandour, Golder Associates Inc. Thursday, April 16.

APPENDIX I
BULKHEAD DESIGN CALCULATIONS

TABLE 1

BULKHEAD SPECIFICATIONS

Bulkhead Location	Elevation (ft)	Maximum Head (ft)	Height (ft)	Width (ft)	Bulkhead Thickness (ft)	Rebar Size	Rebar Spacing (in)	Required Overburden (ft)
Portal #1 Adit	2600	530	17	16	15	#10	8	99
Backfill Crosscut	2633	497	15	16	12.5	#10	8	92
Old Timer's Adits A&B	2928 2928	202 202	15 7	16 5	5 5	#10 #6	7.5 12	38 38

NOTES: Line of sight distances for estimating maximum earthquake loading approximate, but conservative. All concrete to be 3000 psi compressive strength.

LAMEFOOT MINE, PORTAL #1 (Access) ADIT BULKHEAD April 29, 1994

Notation:

a = compression zone depth(in) minimum to balance rebar tension

A_s = area of rebar $\left(\frac{\text{in}^2}{\text{ft}}\right)$

b = beam width (1 in)

C = compressive bending force (lb)

D = dead load $\left(\frac{\text{lb}}{\text{ft}}\right)$

d = distance, extreme compression fiber to rebar centroid (in)

E = bulkhead earthquake load $\left(\frac{\text{lb}}{\text{ft}}\right)$

ΣE_m = total load earthquake load (lb)

FS = factor of safety

$\sqrt{f_c'}$ = square root of f_c'

f_s' = concrete shear strength ($2\sqrt{f'_c}$ psi)

f_y = rebar yield strength (60000 psi)

H = design water head (530 ft)

h = tunnel height (17 ft)

$K = \left(3.5 - 2.5\frac{M_u}{d}\right)$

ℓ = tunnel width (16 ft)

M_n = nominal beam moment (ft·lb)

$M_{u\alpha}$ = earthquake beam moment (ft·lb)

S = section modulus (in^3)

B_p overburden breakdown pressure (psi)

b_w = web width (12 in)

c = centroidal distance (in)

E_m = bulkhead earthquake mass $\left(\frac{\text{lb sec}^2}{\text{ft}}\right)$

F = fluid load $\left(\frac{\text{lb}}{\text{ft}}\right)$

f_c' = concrete comp strength (3000 psi)

f_{ct}' = concrete tensile strength

($3\sqrt{f_c'}$ or $5\phi\sqrt{f_c'}$ psi)

g = acceleration due to gravity ($32.2 \frac{\text{ft}}{\text{sec}^2}$)

H_p = overburden hydrofrac depth (ft)

I = moment of inertia (in^4)

L = beam length or depth (15 ft)

M = bending moment (ft·lb)

M_u = factored beam moment (ft·lb)

m_c = minimum cover, form to rebar (in)

S_ℓ = line-of-sight distance (ft)

T = tensile bending force (lb)

U = required strength $\left(\frac{\text{lb}}{\text{ft}}\right)$

U_α = earthquake required strength $\left(\frac{\text{lb}}{\text{ft}}\right)$

V_c = concrete shear strength (lb)

V_n = nominal shear force (lb)

V_s = rebar shear strength (lb)

V_u = factored shear force (lb)

v_s = rebar shear stress (psi)

W = bulkhead load (lb)

ω = uniform load $\left(\frac{\text{lb}}{\text{ft}}\right)$

α = earthquake acceleration $\left(0.087\frac{\text{ft}}{\text{sec}^2}\right)$

ρ = pressure head (psi)

ρ_g = pressure gradient $\left(\frac{\text{psi}}{\text{ft}}\right)$

$\rho_w = \frac{A_s}{b_w d}$

ϕ = strength reduction factors

0.90 flexure rebar tension

0.85 concrete shear

0.65 plain concrete

γ_w = water density (62.4PCF)

γ_c = concrete density (151PCF)

γ_r = rock density (168 PCF)

σ = flexure stress (psi)

Strength reduction factors (Reinforced Concrete ACI 318-89, rev 1992, Sec 9.3.2.2 & Sec 9.3.2.3; Plain Concrete ACI 318.1-89, rev 1992, Sec 6.2.2)

Load factors (Reinforced Concrete ACI 318-89, rev 1992, Sec 9.2.2, 9.2.3, 9.2.5)

Static fluid load factor (F) = 1.4;
Factor for fluid load under earthquake acceleration (F) = 1.05;
Earthquake accelerated load factor (E) = 1.40

Hydraulic pressure gradient:

Low pressure grouting of concrete-rock contact but not rock, gradient allowable = 41 psi/ft (Garrett & Campbell-Pitt, 1958, Chekan, 1985, p11), with factor of safety of 4

Portal #1 (Access) Adit bulkhead, maximum pressure head

$$\rho = \frac{H\gamma_w}{144} = \frac{530(62.4)}{144} = 230 \text{ psi}$$

Required bulkhead length $\mathbf{L} = \frac{\rho}{41} = \frac{230}{41} = \mathbf{5.6\ ft}$

Pressure gradient with L = 15 ft $\rho_g = \frac{\rho}{L} = \frac{230}{15} = 15.3\ \frac{\text{psi}}{\text{ft}}$ $\mathbf{FS} = \frac{41}{15.3} = \mathbf{2.68}$

Concrete shear on Portal #1 (Access) Adit perimeter:

$$f'_s = 2\sqrt{f'_c} = 2\sqrt{3000} = 110 \text{ psi} \qquad \text{(ACI 318-89, rev 1992, Sec 11.3.1.1)}$$

$$L = \frac{\rho h\ell}{[2(h+\ell)f'_s]} = \frac{230(17)16}{[2(17+16)110]} = \mathbf{8.6\ ft}$$

$$W = \rho h\ell = 230(17)16(144) = 90010000 \text{ lb}$$

$$v_s = \frac{W}{[2(h+\ell)]L(144)} = \frac{9000000}{[2(17+16)]15(144)} = 63 \text{ psi}$$

$$\mathbf{FS} = \frac{110}{63} = \mathbf{1.75}$$

Plain concrete deep-beam bending stress design, Portal #1 (Access) Adit (ACI 318-89, rev 1992), Sec 9.2.5 & 18.4.1(b), & ACI 318.1-89, rev 1992, Sec 6.2.2)

$$\omega = U = 1.4\rho(144) = 1.4(230)144 = 46400 \frac{\text{lb}}{\text{ft}}$$

$$M_n = \frac{\omega\ell^2}{8} = \frac{46400(16^2)}{8} = 1480000 \text{ ft}\cdot\text{lb}$$

$$M_u = \frac{M_n}{0.65} = \frac{1480000}{0.65} = 2280000 \text{ ft}\cdot\text{lb}$$

$$S = \frac{I}{c} = \frac{\frac{bL^3}{12}}{\frac{L}{2}} = \frac{\frac{1(L^3)(12^3)}{12}}{\frac{L(12)}{2}} = \frac{144L^2}{6}$$

$$f'_{cl} = 3\sqrt{f'_c} = 3\sqrt{3000} = 164 \text{ psi (ACI 318-89, rev 1992, Sec 18.4.1(b))}$$

$$f'_{cl} = 5\phi\sqrt{f'_c} = 5(0.65)\sqrt{3000} = 178 \text{ psi}$$ (ACI 318.1-89, rev 1992, Sec 6.2.1(a) & 6.2.2)

$$f'_{cl} = 164 = \sigma = \frac{M_u c}{I} = \frac{M_u}{S} = \frac{2280000}{\frac{144L^2}{6}} = \frac{95000}{L^2}$$

$$L = \sqrt{\frac{95800}{164}} = \mathbf{24.1\ ft}$$ **∴ 15-ft long bulkhead must be reinforced**

Reinforced concrete deep-beam bending stress design, Portal #1 (Access) Adit (ACI 318-89, Sec 9.3.2.1, Sec 10.5.1, Sec 11.8. Wang & Salmon, 1985, Einarson & Abel, 1990)

$$C = \phi f'_c b_w a = 0.85(3000)12a = 30600a$$

$$T = A_s f_y = 60000A_s$$

$$C = T\ ;\ a = \frac{60000A_s}{30600} = 1.961A_s$$

$$M_n = \frac{\omega \ell^2}{8} = \frac{46400(16^2)}{8} = 1480000 \text{ ft} \cdot \text{lb}$$

$$M_u = \frac{M_n}{\phi} = \frac{1480000}{0.9} = 1640000 \text{ ft} \cdot \text{lb } (19700000 \text{ in} \cdot \text{lb})$$

$$M_u = A_s f_y\left(d - \frac{a}{2}\right);\ d = L - m_c = 15(12) - 3 = 177 \text{ in}$$

$$M_u = 60000A_s\left(117 - \frac{a}{2}\right) = 60000A_s\left(177 - \frac{1.961A_s}{2}\right) = -58800A_s^2 + 10600000A_s$$

$$\therefore\ 19700000 = -58800A_s^2 + 10600000A_s$$

$$58800A_s^2 - 10600000A_s + 19700000 = 0$$

$$\mathbf{A_s = 1.88\ \frac{in^2}{ft}}$$ **steel area required**

#10 bars on 8-in c-c provides 1.91 $\frac{\text{in}^2}{\text{ft}}$ steel area

Check for adequacy

Allowable $M_u = -58800A_s^2 + 10600000A_s = 20000000 \text{ in}\cdot\text{lb}$

Design $M_u = 19700000 \text{ in}\cdot\text{lb}$

$$\mathbf{FS} = \frac{20000000}{19700000} = \mathbf{1.02}$$

Critical section shear strength for Portal #1 (Access) Adit, 15-ft deep beam bulkhead

Deep beam defined as $\frac{\ell}{d} < 5$ (ACI 318-89 rev 1992, Sec 11.8.1) $\frac{\ell}{d} = \frac{16}{15} = 1.1 < 5.$

$\therefore$ bulkhead is a deep beam for design

Critical section shear at 0.15ℓ (2.40 ft) from ribside (ACI 318-89, Sec 11.8.5), with #10 bars on 8 in c-c, there will be 1.91 $\frac{\text{in}^2}{\text{ft}}$ of steel per ft of beam width, d = 177 in (14.75 ft).

Detailed shear strength at critical section (ACI 318-89 rev 1992, Sec 11.8.7)

$$V_c = K\left(1.9\sqrt{f_c'} + 2500\rho\frac{V_u d}{M_u}\right)b_w d$$

$$M_n = \left(\frac{\omega\ell}{2}\right)(0.15\ell) - \omega(0.15\ell)\frac{0.15\ell}{2} = 0.06375\omega\ell^2$$

$$M_n = 0.06375\left[46400\left(16^2\right)\right] = 757000 \text{ ft}\cdot\text{lb}$$

$$M_u = \frac{M_n}{0.9} = \frac{757000}{0.9} = 841000 \text{ ft } M_u = \frac{M_n}{0.9} = \frac{757000}{0.9} = 841000 \text{ ft}\cdot\text{lb}$$

$$V_n = \frac{\omega\ell}{2} - \left(\frac{\omega\ell}{2}\right)\left(\frac{0.15\ell}{0.5\ell}\right) = 0.35\omega\ell = 0.35(46400)16 = 260000 \text{ lb}$$

$$V_u = \frac{V_n}{0.85} = \frac{260000}{0.85} = 306000 \text{ lb}$$

$$K = 3.5 - 2.5\frac{M_u}{V_u d} = 3.5 - 2.5\left[\frac{841000}{306000\left(\frac{177}{12}\right)}\right] = 3.5 - 0.47 = 3.03$$

K cannot exceed 2.5 $\therefore K = 2.5$

$$\rho_w = \frac{A_s}{b_w d} = \frac{1.91}{(12)177} = 0.00090$$

$$V_c = K\left(1.9\sqrt{f_c'} + 2500\rho_w \frac{V_u d}{M_u}\right)bd$$

$$V_c = 2.5\left[1.9\sqrt{3000} + 2500(0.00090)\frac{306000(14.75)}{821000}\right]12(177)$$

$$V_c = 2.5[104.1 + 12.4]2124 = 2.5[116.5]2124 = 619000 \text{ lb}$$

$$\mathbf{FS} = \frac{V_c}{V_u} = \frac{619000}{306000} = \mathbf{2.02}$$

Earthquake bulkhead design; Load factors (ACI 318 rev 1992, Sec 9.2.2, 9.2.3, 9.2.5) Factor for fluid load under earthquake acceleration (F) = 1.05; Load factor for earthquake accelerated mass (E) = 1.40

$U = 1.05F + 1.40E$

Factored fluid load = $1.05F = 1.05(230)144 = 34800$ lb/ft

Mass accelerated by earthquake (E)

$$E_m = \frac{S_\ell \gamma_w h \ell + L h \ell \gamma_c}{g} = \frac{1050(62.4)17(16) + 15(17)16(151)}{32.2}$$

$$E_m = \frac{[17800000 + 616000]}{32.2} = \frac{18400000}{32.2} = 572000 \ \frac{\text{lb}\cdot\text{sec}^2}{\text{ft}}$$

$$\Sigma E_m = E_m \alpha = 572000(0.016) = 9150 \text{ lb}$$

$$E = \frac{\Sigma E_m}{h} = \frac{9150}{17} = 538 \ \frac{\text{lb}}{\text{ft}}$$

Total load under earthquake acceleration

$$\rho = \frac{H\gamma_w}{144} = \frac{530(62.4)}{144} = 230 \text{ psi}$$

$$F = \rho b_w(12) = 230(12)12 = 33100 \frac{\text{lb}}{\text{ft}}$$

$$U_\alpha = 1.05F + 1.40E = 1.05(33100) + 1.40(538) = 34800 + 753 = 35600 \frac{\text{lb}}{\text{ft}}$$

Earthquake nominal beam bending moment

$$M_{n\alpha} = \frac{U_\alpha \ell}{8} = \frac{35600(16^2)}{8} = 1140000 \text{ ft} \cdot \text{lb}$$

$$M_{u\alpha} = \frac{M_{n\alpha}}{0.9} = \frac{1140000}{0.9} = 1270000 \text{ ft} \cdot \text{lb } (15200000 \text{ in} \cdot \text{lb})$$

Steel area required for earthquake loading

$$588800A_s^2 - 10600000A_s + 15200000$$

$$A_s = 1.45 \frac{\text{in}^2}{\text{ft}} \text{ steel area required}$$

#10 bars on 8-in c-c provide 1.91 $\frac{\text{in}^2}{\text{ft}}$ steel area

Check for adequacy

$$\text{Allowable } M_{u\alpha} = -58800A_s^2 + 10600000A_s = 20000000 \text{ in} \cdot \text{lb}$$

$$\text{Design } M_{u\alpha} = 15200000 \text{ in} \cdot \text{lb}$$

$$\mathbf{FS} = \frac{20000000}{15200000} = \mathbf{1.32}$$

Overburden depth (H_r) required to prevent hydrofracturing of adjacent rock (Einarson & Abel, 1990, p 71-15)

$$\mathbf{H_r} = \frac{72B_p}{\gamma_r} = \frac{72(230)}{168} = \mathbf{99\ ft}$$

John F. Abel, Jr.
Colorado P.E. 5642

LAMEFOOT MINE, BACKFILL CROSSCUT BULKHEAD

April 30, 1994

Notation:

a = compression zone depth(in) minimum to balance rebar tension

A_s = area of rebar $\left(\frac{in^2}{ft}\right)$

B_p overburden breakdown pressure (psi)

b = beam width (1 in)

b_w = web width (12 in)

C = compressive bending force (lb)

c = centroidal distance (in)

D = dead load $\left(\frac{lb}{ft}\right)$

d = distance, extreme compression fiber to rebar centroid (in)

E = bulkhead earthquake load $\left(\frac{lb}{ft}\right)$

E_m = bulkhead earthquake mass $\left(\frac{lb \cdot sec^2}{ft}\right)$

ΣE_m = total load earthquake load (lb)

F = fluid load $\left(\frac{lb}{ft}\right)$

FS = factor of safety

f'_c = concrete comp strength (3000 psi)

$\sqrt{f'_c}$ = square root of f'_c

f'_{ct} = concrete tensile strength

($3\sqrt{f'_c}$ or $5\phi\sqrt{f'_c}$ psi)

f'_s = concrete shear strength ($2\sqrt{f'_c}$ psi)

f_y = rebar yield strength (60000 psi)

g = acceleration due to gravity ($32.2 \frac{ft}{sec^2}$)

H = design water head (497 FT)

H_p = overburden hydrofrac depth (ft)

h = tunnel height (15 ft)

I = moment of inertia $\left(in^4\right)$

$K = \left(3.5 - 2.5\frac{M_u}{d}\right)$

L = beam length or depth (12.5 ft)

ℓ = tunnel width (16 ft)

M = bending moment (ft·lb)

M_n = nominal beam moment (ft·lb)

M_u = factored beam moment (ft·lb)

$M_{u\alpha}$ = earthquake beam moment (ft·lb)

m_c = minimum cover, form to rebar (in)

S = section modulus $\left(in^3\right)$

S_ℓ = line-of-sight distance (ft)

T = tensile bending force (lb)

U = required strength $\left(\frac{\text{lb}}{\text{ft}}\right)$

U_α = earthquake required strength $\left(\frac{\text{lb}}{\text{ft}}\right)$

V_c = concrete shear strength (lb)

V_n = nominal shear force (lb)

V_s = rebar shear strength (lb)

V_u = factored shear force (lb)

v_s = rebar shear stress (psi)

W = bulkhead load (lb)

ω = uniform load $\left(\frac{\text{lb}}{\text{ft}}\right)$

α = earthquake acceleration $\left(0.087\frac{\text{ft}}{\text{sec}^2}\right)$

ρ = pressure head (psi)

ρ_g = pressure gradient $\left(\frac{\text{psi}}{\text{ft}}\right)$

$\rho_w = \frac{A_s}{b_w d}$

ϕ = strength reduction factors

0.90 flexure rebar tension

0.85 concrete shear

0.65 plain concrete

γ_w = water density (62.4PCF)

γ_c = concrete density (151PCF)

γ_r = rock density (168 PCF)

σ = flexure stress (psi)

Strength reduction factors (Reinforced Concrete ACI 318-89, rev 1992, Sec 9.3.2.2 & Sec 9.3.2.3; Plain Concrete ACI 318.1-89, rev 1992, Sec 6.2.2)

Load factors (Reinforced Concrete ACI 318-89, rev 1992, Sec 9.2.2, 9.2.3, 9.2.5)

Static fluid load factor (F) = 1.4;
Factor for fluid load under earthquake acceleration (F) = 1.05;
Earthquake accelerated load factor (E) = 1.40

Hydraulic pressure gradient:

Low pressure grouting of concrete-rock contact but not rock, gradient allowable = 41 psi/ft (Garrett & Campbell-Pitt, 1958, Chekan, 1985, p11), with factor of safety of 4

Backfill Crosscut bulkhead, maximum pressure head

$$\rho = \frac{H\gamma_w}{144} = \frac{497(62.4)}{144} = 215 \text{ psi}$$

Required bulkhead length $\mathbf{L} = \frac{\rho}{41} = \frac{215}{41} = \mathbf{5.2\ ft}$

Pressure gradient with L = 12.5 ft $\rho_g = \frac{\rho}{L} = \frac{215}{12.5} = 17.2\ \frac{\text{psi}}{\text{ft}}$ $\mathbf{FS} = \frac{41}{17.2} = \mathbf{2.38}$

Concrete shear on Backfill Crosscut perimeter:

$$f'_s = 2\sqrt{f'_c} = 2\sqrt{3000} = 110 \text{ psi} \qquad \text{(ACI 318-89, rev 1992, Sec 11.3.1.1)}$$

$$L = \frac{ph\ell}{\left[2(h+\ell)f'_s\right]} = \frac{215(15)16}{\left[2(15+16)110\right]} = \mathbf{7.57\ ft}$$

$$W = ph\ell = 215(15)16(144) = 7430000 \text{ lb}$$

$$v_s = \frac{W}{\left[2(h+\ell)\right]L(144)} = \frac{7430000}{\left[2(15+16)\right]12.5(144)} = 66.6 \text{ psi}$$

$$\mathbf{FS} = \frac{110}{66.6} = \mathbf{1.65}$$

Plain concrete deep-beam bending stress design, Backfill Crosscut (ACI 318-89, rev 1992), Sec 9.2.5 & 18.4.1(b), & ACI 318.1-89, rev 1992, Sec 6.2.2)

$$\omega = U = 1.4p(144) = 1.4(215)144 = 43300 \frac{\text{lb}}{\text{ft}}$$

$$M_n = \frac{\omega \ell^2}{8} = \frac{43300(15^2)}{8} = 1220000 \text{ ft} \cdot \text{lb}$$

$$M_u = \frac{M_n}{0.65} = \frac{1220000}{0.65} = 1880000 \text{ ft} \cdot \text{lb}$$

$$S = \frac{I}{c} = \frac{\frac{bL^3}{12}}{\frac{L}{2}} = \frac{\frac{1(L^3)(12^3)}{12}}{\frac{L(12)}{2}} = \frac{144L^2}{6}, \text{ Einarson \& Abel, 1990)}$$

$$f'_{ct} = 3\sqrt{f'_c} = 3\sqrt{3000} = 164 \text{ psi (ACI 318-89, rev 1992, Sec 18.4.1(b))}$$

$$f'_{ct} = 5\phi\sqrt{f'_c} = 5(0.65)\sqrt{3000} = 178 \text{ psi (ACI 318.1-89, rev 1992, Sec 6.2.1(a) \& 6.2.2)}$$

$$f'_{ct} = 164 = \sigma = \frac{M_u c}{I} = \frac{M_u}{S} = \frac{1880000}{\frac{144L^2}{6}} = \frac{78300}{L^2}$$

$$L = \sqrt{\frac{78300}{164}} = \mathbf{21.9ft}$$

∴ 12.5-ft long bulkhead must be reinforced

Reinforced concrete deep-beam bending stress design, Backfill Crosscut (ACI 318-89, Sec 9.3.2.1, Sec 10.5.1, Sec 11.8. Wang & Salmon, 1985

$C = \phi f'_c b_w a = 0.85\ 3000 \qquad \phi 12a = 30600a$

$T = A_s f_y = 60000A_s$

$C = T\ ;\ a = \frac{60000A_s}{30600} = 1.961A_s$

$M_n = \frac{\omega\ ^2}{8} = \frac{43300\ 15^2}{8} = 1220000\ \text{ft}\cdot\text{lb}$

$M_u = \frac{M_n}{\phi} = \frac{1220000}{0.9} = 1360000\ \text{ft}\cdot\text{lb}\ \ 16300000\ \text{in}\cdot\text{lb} \qquad \phi$

$M_u = A_s f_y\ d - \frac{a}{2} \qquad ;\ d = L - m_c = 12.5\ 12 \qquad \phi - 3 = 147\ \text{in}$

$M_u = 60000A_s\ 141 - \frac{a}{2} \qquad = 60000A_s\ 147 - \frac{1.961A_s}{2} \qquad = -58800A_s^2 + 8820000A_s$

$\therefore\ 16300000 = -58800A_s^2 + 8820000A_s$

$58800A_s^2 - 8820000A_s + 16300000 = 0$

$\mathbf{A_S = 1.87\ \frac{in^2}{ft}}$ **steel area required**

#10 bars on 8-in c-c provide 1.91 $\frac{\text{in}^2}{\text{ft}}$ **steel area**

Check for adequacy

Allowable $M_u = -58800A_s + 8820000A_s = 16600000\text{in}\cdot\text{lb}$

Design $M_u = 16300000\text{in}\cdot\text{lb}$

$\mathbf{FS} = \frac{16600000}{16300000} = \mathbf{1.02}$

Critical section shear strength for Portal #1 (Access) Adit, 15-ft deep beam bulkhead

Deep beam defined as $\frac{\ell}{d} < 5$ (ACI 318-89 rev 1992, Sec 11.8.1) $\frac{\ell}{d} = \frac{15}{12.5} = 1.2 < 5.$

$\therefore$ bulkhead is a deep beam for design

Critical section shear at 0.15ℓ (2.25 ft) from ribside (ACI 318-89, rev 1992, Sec 11.8.5), with #10

bars on 8 in c-c, there will be $1.91\ \frac{\text{in}^2}{\text{ft}}$ of steel per ft of beam width, d = 177 in (14.75 ft).

Detailed shear strength at critical section (ACI 318-89, rev 1992, Sec 11.8.7)

$$V_c = K\left(1.9\sqrt{f'_c} + 2500\rho_w \frac{V_u d}{M_u}\right) b_w d$$

$$M_n = \frac{\omega\ell}{2}(0.15\ell) - \omega(0.15\ell)\frac{0.15\ell}{2} = 0.06375\omega\ell^2$$

$$M_n = 0.06375\left[43300\left(15^2\right)\right] = 621000\text{ft}\cdot\text{lb}$$

$$M_u = \frac{M_n}{0.9} = \frac{621000}{0.9} = 690000\text{ft}\cdot\text{lb}$$

$$V_n = \frac{\omega\ell}{2} - \left(\frac{\omega\ell}{2}\right)\left(\frac{0.15\ell}{0.5\ell}\right) = 0.35\omega\ell$$

$$V_n = \frac{\omega\ell}{2} - \left(\frac{\omega\ell}{2}\right)\left(\frac{0.15\ell}{0.5\ell}\right) = 0.35\omega\ell = 0.35(43300)15 = 227000\text{lb}$$

$$V_u = \frac{V_n}{0.85} = \frac{227000}{0.85} = 267000\text{lb}$$

$$K = 3.5 - 2.5\frac{M_u}{V_u d} = 3.5 - 2.5\left[\frac{690000}{227000\left(\frac{147}{12}\right)}\right] = 3.5 - 0.62 = 2.88$$

K cannot exceed 2.5 $\therefore K = 2.5$

$$\rho_w = \frac{1.91}{(12)147} = 0.00108$$

$$V_c = K\left(1.9\sqrt{f_c'} + 2500\rho\frac{V_u d}{M_u}\right)b_w d$$

$$V_c = 2.5\left[1.9\sqrt{3000} + 2500(0.00108)\frac{267000(12.25)}{346000}\right]12(147)$$

$$V_c = 2.5[104.1 + 25.5]1764 = 2.5[129.6]1764 = 572000\text{lb}$$

$$\mathbf{FS} = \frac{V_c}{V_u} = \frac{572000}{267000} = \mathbf{2.14}$$

Earthquake bulkhead design; Load factors (ACI 318 rev 1992, Sec 9.2.2, 9.2.3, 9.2.5) Factor for fluid load under earthquake acceleration (F) = 1.05; Load factor for earthquake accelerated mass (E) = 1.40

U = 1.05F + 1.40E

Factored fluid load = 1.05F =1.05(215)144 = 32500 lb/ft

Mass accelerated by earthquake (E)

$$E_m = \frac{S_\ell \gamma_w h\ell + Lh\ell\gamma_c}{32.2} = \frac{[500(62.4)16(15) + 12.5(16)15(151)]}{32.2}$$

$$E_m = \frac{[7490000 + 453000]}{32.2} = \frac{[7940000]}{32.2} = 247000\frac{\text{lb}\cdot\text{sec}^2}{\text{ft}}$$

$$\Sigma E_m = E_m\alpha = 247000(0.016) = 3952\text{lb}$$

$$E = \frac{\Sigma E_m}{\ell} = \frac{3952}{16} = 247\frac{\text{lb}}{\text{ft}}$$

Total load under earthquake acceleration

$$\rho = \frac{H(62.4)}{144} = \frac{497(62.4)}{144} = 215\text{psi}$$

$$F = \rho b_w(12) = 215(12)12 = 31000\frac{\text{lb}}{\text{ft}}$$

$$U_\alpha = 1.05F + 1.40E = 1.05(31000) + 1.40(247) = 32600 + 346 = 32900\frac{\text{lb}}{\text{ft}}$$

Earthquake nominal beam bending moment

$$M_{n\alpha} = \frac{U_\alpha h}{8} = \frac{32900(15^2)}{8} = 925000\ \text{ft}\cdot\text{lb}$$

$$M_{u\alpha} = \frac{M_{n\alpha}}{0.9} = \frac{925000}{0.9} = 1030000\ \text{ft}\cdot\text{lb}\ (12400000\ \text{in}\cdot\text{lb})$$

Steel area required for earthquake loading

$$58800A_s^2 - 8820000A_s + 12400000$$

$$A_s = 1.42\ \frac{\text{in}^2}{\text{ft}}$$ steel area required

#10 bars on 8-in c-c provide 1.91 $\frac{\text{in}^2}{\text{ft}}$ steel area

Check for adequacy

Allowable $M_{u\alpha} = -58800A_s^2 + 8820000A_s = 16600000\ \text{in}\cdot\text{lb}$

Design $M_{u\alpha} = 12400000\ \text{in}\cdot\text{lb}$

$$\mathbf{FS} = \frac{16600000}{12400000} = \mathbf{1.34}$$

Overburden depth (H_r) required to prevent hydrofracturing of adjacent rock (Einarson & Abel, 1990, p 71-15)

$$\mathbf{H_r} = \frac{72B_p}{\gamma_r} = \frac{72(215)}{168} = \mathbf{92\ ft}$$

John F. Abel, Jr.
Colorado P.E. 5642

OLD TIMER'S ADIT BULKHEAD DESIGN CALCULATIONS, 15 x 16 ft

April 30, 1994

Notation:

a = compression zone depth(in) minimum to balance rebar tension

A_s = area of rebar $\left(\frac{in^2}{ft}\right)$

b = beam width (1 in)

C = comp bending force (lb)

D = dead load $\left(\frac{lb}{ft}\right)$

F = fluid load $\left(\frac{lb}{ft}\right)$

FS = factor of safety

$\sqrt{f'_c}$ = square root of f'_c

f'_r = rock cohesion (>210 psi)

f_y = rebar yield strength (60000 psi)

h = tunnel height (15 ft)

$K = \left(3.5 - 2.5\frac{M_u}{d}\right)$

ℓ = tunnel width (16 ft)

M_n = nominal beam moment (ft·lb)

m_c = minimum cover, form face to rebar surface (3 in)

s = distance of beam analysis (12 in)

U = required strength $\left(\frac{lb}{ft}\right)$

V_n = nominal shear force (lb)

V_u = factored shear force (lb)

v_s = rebar shear stress (psi)

ω = uniform load $\left(\frac{lb}{ft}\right)$

A_v = area of shear reinforcement(in^2)

b_w = web width (12 in)

c = centroidal distance (in)

d = distance, extreme compression fiber to rebar centroid

f'_c = concrete comp strength (3000 psi)

f'_{ct} = concrete tensile strength (psi)

f'_s = concrete shear strength ($2\sqrt{f'_c}$ psi)

H = design water head (202 ft)

I = moment of inertia (in^4)

L = bulkhead thickness or depth

M = bending moment (ft·lb)

M_u = factored beam moment (ft·lb)

ρ = maximum pressure head (88 psi)

S = section modulus (in^3)

T = tensile bending force (lb)

V_c = concrete shear strength (lb)

V_s = rebar shear strength (lb)

v_c = concrete shear stress (psi)

W = bulkhead load (lb)

ρ_g = pressure gradient $\left(\frac{psi}{ft}\right)$

ρ = ratio of nonprestressed tensile reinforcement to concrete compressivearea = $\left(\frac{A_s}{b_w d}\right)$

strength reduction factors

0.90 flexure rebar tension

0.85 concrete shear

0.65 plain concrete flexure

γ_w = water density (62.4PCF)

γ_c = concrete density (151PCF)

γ_r = rock density (168 PCF)

σ = flexure stress (psi)

Strength reduction factors (Reinforced Concrete ACI 318-89, rev 1992, Sec 9.3.2.2 & Sec 9.3.2.3; Plain Concrete, ACI 318.1-89, rev 1992, Sec 6.2.2)

Load factors (ACI 318R-92, Sec 9.2.2, 9.2.3, 9.2.5)

Static fluid load factor (F) = 1.4
Factor for fluid load under earthquake acceleration (F) = 1.05
Earthquake accelerated load factor (E) = 1.40

Hydraulic pressure gradient:

Low pressure grouting of concrete-rock contact but not rock, gradient allowable = 41 psi/ft (Garrett & Campbell-Pitt, 1958, Chekan, 1985, p11), with factor of safety of 4

Old Timer's Adit bulkheads, maximum pressure head

$$p = \frac{H\gamma_w}{144} = \frac{202(62.4)}{144} = 88 \text{ psi}$$

Required bulkhead thickness $\mathbf{L} = \frac{p}{41} = \frac{88}{41} = \mathbf{2.15\ ft}$

Pressure gradient with L = 5 ft $\quad p_g = \frac{p}{L} = \frac{88}{5} = 18\ \frac{\text{psi}}{\text{ft}} \qquad \textbf{FS} = \frac{41}{18} = \mathbf{2.28}$

Concrete shear, Old Timer's Adit bulkhead perimeter, 15 x 16 ft:

$f'_s = 2\sqrt{f'_c} = 2\sqrt{3000} = 110$ psi (ACI 318R-89, rev 1992, Sec 11.3.1.1)

$$\mathbf{L} = \frac{\phi p h \ell}{[2(h+\ell)f'_s]} = \frac{(1.4)88(15)16}{[2(15 + 16)110]} = \mathbf{4.34\ ft}$$

$$W = \phi p h \ell = (1.4)88(15)16(144) = 4260000 \text{ lb}$$

$$v_c = \frac{W}{[2(h+\ell)]L(144)} = \frac{4260000}{[2(15+16)]5(144)} = 95.4 \text{ psi}$$

$$\mathbf{FS} = \frac{110}{95.4} = \mathbf{1.15}$$

Rock shear, Old Timer's Adit bulkhead perimeter, 15 x 16 ft:

$f'_r \geq 210$ psi (minimum cohesion from physical test results)

$$L = \frac{\phi p h \ell}{\left[2(h+\ell)f'_r\right]} = \frac{(1.4)88(15)16}{\left[2(15\ +\ 16)210\right]} = \mathbf{<2.27\ ft}$$

$$W = \phi p h \ell = (1.4)88(15)16(144) = 4260000 \text{ lb}$$

$$v_s = \frac{W}{\left[2(h+\ell)\right]L(144)} = \frac{4260000}{\left[2(15+16)\right]5(144)} = 95.4 \text{ psi}$$

$$\mathbf{FS} \geq \frac{210}{95.4} \geq \mathbf{2.20}$$

Plain concrete deep-beam bending stress design, Old Timer's Adit bulkhead perimeter, 15 x 16 ft: (ACI 318R-89, rev 1992, Sec 9.9.2.5, 18.4.1(b), & ACI 318.1-89, rev 1992, Sec 6.2.2)

$$\omega = U = \phi p(144) = 1.4(88)144 = 17700\ \frac{\text{lb}}{\text{ft}}$$

$$M_n = \frac{\omega h^2}{8} = \frac{17700(15^2)}{8} = 498000 \text{ ft}\cdot\text{lb}$$

$$M_u = \frac{M_n}{0.65} = \frac{498000}{0.65} = 766000 \text{ ft}\cdot\text{lb}$$

$$S = \frac{I}{c} = \frac{\frac{bL^3}{12}}{\frac{L}{2}} = \frac{\frac{1(L^3)(12^3)}{12}}{\frac{L(12)}{2}} = \frac{144L^2}{6}$$

$$f'_{cl} = 3\sqrt{f'_c} = 3\sqrt{3000} = 164 \text{ psi (ACI 318-89, rev 1992, Sec 18.4.1(b))}$$

$$f'_{cl} = 5\phi\sqrt{f'_c} = 5(0.65)\sqrt{3000} = 178 \text{ psi (ACI 318.1-89, rev 1992, Sec 6.2.1(a) \& 6.2.2)}$$

$$f'_{cl} = 164 = \sigma = \frac{M_u c}{I} = \frac{M_u}{S} = \frac{766000}{\frac{144L^2}{6}} = \frac{31900}{L^2}$$

$$L = \sqrt{\frac{31900}{164}} = \mathbf{14.0\ ft}$$

∴ 5-ft long bulkhead must be reinforced

Reinforced concrete beam bending stress design, Old Timer's Adit bulkhead perimeter, 15 x 16 ft: (ACI 318-89, rev 1992, Sec 9.3.2.1, Sec 10.5.1, Sec 11.8. Wang & Salmon, 1985, Einarson & Abel, 1990)

$$C = \phi f_c' b_w a = 0.85(3000)12a = 30600a$$

$$T = A_s f_y = 60000A_s$$

$$C = T\ ;\ a = \frac{60000A_s}{30600} = 1.961A_s$$

$$M_n = \frac{\omega h^2}{8} = \frac{17700(15^2)}{8} = 498000\ \text{ft}\cdot\text{lb}$$

$$M_u = \frac{M_n}{\phi} = \frac{498000}{0.9} = 553000\ \text{ft}\cdot\text{lb}\ \ (6640000\ \text{in}\cdot\text{lb})$$

$$M_u = A_s f_y\left(d - \frac{a}{2}\right);\ d = L - m_c = 5(12) - 3 = 57\ \text{in}$$

$$M_u = 60000A_s\left(57 - \frac{a}{2}\right) = 60000A_s\left(57 - \frac{1.961A_s}{2}\right) = -58800A_s^2 + 3420000A_s$$

$$\therefore\ 6640000 = -58800A_s^2 + 3420000A_s$$

$$58800A_s^2 - 3420000A_s + 6640000 = 0$$

$A_s = 2.01\ \frac{\text{in}^2}{\text{ft}}$ steel area required to support bending

#10 bars on 7.5-in c-c provide 2.03 $\frac{\text{in}^2}{\text{ft}}$ steel area

Check for adequacy

$$\text{Allowable } M_u = -58800A_s^2 + 3420000A_s = 6700000\ \text{in}\cdot\text{lb}$$

$$\text{Design } M_u = 6640000\ \text{in}\cdot\text{lb}$$

$$\mathbf{FS} = \frac{6700000}{6640000} = \mathbf{1.01}$$

Critical section shear strength for Old Timer's Adit, 5-ft thick beam bulkhead

Deep beam defined as $\frac{\ell}{d} < 5$ (ACI 318-89 rev 1992, Sec 11.8.1) $\frac{\ell}{d} = \frac{15}{4.75} = 3.16 < 5.$

∴ bulkhead is a deep beam for design

Detailed shear strength at critical section (ACI 318-89, rev 1992, Sec 11.8.7)

$$M_n = \left(\frac{\omega\ell}{2}\right)(0.15\ell) - \omega(0.15\ell)\frac{(0.15\ell)}{2} = 0.06375\omega\ell^2$$

$$M_n = 0.06375\left[17700\left(15^2\right)\right] = 254000 \text{ ft}\cdot\text{lb}$$

$$M_u = \frac{M_n}{0.9} = \frac{254000}{0.9} = 282000 \text{ ft}\cdot\text{lb}$$

$$V_n = \frac{\omega\ell}{2} - \left(\frac{\omega\ell}{2}\right)\left(\frac{0.15\ell}{0.5\ell}\right) = 0.35\omega\ell = 0.35(17700)15 = 92000 \text{ lb}$$

$$V_u = \frac{V_n}{0.85} = \frac{92900}{0.85} = 109000 \text{ lb}$$

$$K = 3.5 - 2.5\frac{M_u}{V_u d} = 3.5 - 2.5\frac{282000}{109000\left(\frac{57}{12}\right)} = 3.5 - 1.36 = 2.14$$

K cannot exceed 2.5 $\therefore K = 2.14$

$$\rho = \frac{A_s}{b_w d} = \frac{2.03}{(12)57} = 0.00297$$

$$V_c = K\left(1.9\sqrt{f_c'} + 2500\rho\frac{V_u d}{M_u}\right)b_w d$$

$$V_c = 2.14\left[1.9\sqrt{3000} - 2500(0.00297)\frac{109000(4.75)}{141000}\right]12(57)$$

$$V_c = 2.14[104.1 - 27.3]684 = 2.14[131.4]684 = 192000 \text{ lb}$$

$$\mathbf{FS} = \frac{V_c}{V_u} = \frac{192000}{109000} = \mathbf{1.76}$$

Earthquake bulkhead design; Load factors (ACI 318 rev 1992, Sec 9.2.2, 9.2.3, 9.2.5) Factor for fluid load under earthquake acceleration (F) = 1.05; Load factor for earthquake accelerated mass (E) = 1.40

$$U = 1.05F + 1.40E$$

Factored fluid load = $1.05F = 1.05(88)144 = 13300$ lb/ft

Mass accelerated by earthquake (E)

$$E_m = \frac{S_\ell \gamma_w h\ell + Lh\ell\gamma_c}{32.2} = \frac{[500(62.4)16(15) + 5(16)15(151)]}{32.2}$$

$$E_m = \frac{[7490000 + 184000]}{32.2} = \frac{[7670000]}{32.2} = 238000 \frac{\text{lb}\cdot\text{sec}^2}{\text{ft}}$$

$$\Sigma E_m = E_m\alpha = 238000(0.016) = 3808 \text{ lb}$$

$$E = \frac{\Sigma E_m}{\ell} = \frac{3808}{16} = 238 \frac{\text{lb}}{\text{ft}}$$

Total load under earthquake acceleration

$$p = \frac{H\gamma_w}{144} = \frac{202(62.4)}{144} = 88 \text{ psi}$$

$$F = pb_w(12) = 88(12)12 = 12700 \frac{\text{lb}}{\text{ft}}$$

$$U_\alpha = 1.05F + 1.40E = 1.05(12700) + 1.40(238) = 13300 + 333 = 13600 \frac{\text{lb}}{\text{ft}}$$

Earthquake nominal beam bending moment

$$M_{n\alpha} = \frac{U_\alpha h^2}{8} = \frac{13600(15^2)}{8} = 383000 \text{ ft}\cdot\text{lb}$$

$$M_{u\alpha} = \frac{M_{n\alpha}}{0.9} = \frac{383000}{0.9} = 426000 \text{ ft}\cdot\text{lb } (5110000 \text{ in}\cdot\text{lb})$$

Steel area required for earthquake loading

$$588000A_s^2 - 3420000A_s + 5110000 = 0$$

$A_s = 1.53 \frac{\text{in}^2}{\text{ft}}$ steel area required to withstand design earthquake loading

#10 bars on 7.5 in c-c provide 2.03 $\frac{\text{in}^2}{\text{ft}}$ steel area

OLD TIMER'S ADIT BULKHEAD DESIGN CALCULATIONS, 7 x 5 ft

April 30, 1994

Notation:

a = compression zone depth(in) minimum to balance rebar tension

A_s = area of rebar $\left(\frac{in^2}{ft}\right)$

b = beam width (1 in)

C = comp bending force (lb)

D = dead load $\left(\frac{lb}{ft}\right)$

F = fluid load $\left(\frac{lb}{ft}\right)$

FS = factor of safety

$\sqrt{f'_c}$ = square root of f'_c

f'_r = rock cohesion (>210 psi)

f_y = rebar yield strength (60000 psi)

h = tunnel height (7 ft)

$K = \left(3.5 - 2.5\frac{M_u}{d}\right)$

l = tunnel width (5 ft)

M_n = nominal beam moment (ft·lb)

m_c = minimum cover, form face to rebar surface (3 in)

s = distance of beam analysis (12 in)

U = required strength $\left(\frac{lb}{ft}\right)$

V_n = nominal shear force (lb)

V_u = factored shear force (lb)

v_s = rebar shear stress (psi)

w = uniform load $\left(\frac{lb}{ft}\right)$

A_v = area of shear reinforcement(in)

b_w = web width (12 in)

c = centroidal distance (in)

d = distance, extreme compression fiber to rebar centroid

f' = concrete comp strength (3000 psi)

f'_{ct} = concrete tensile strength (psi)

f'_s = concrete shear strength ($2\sqrt{f'_c}$ psi)

H = design water head (202 ft)

I = moment of inertia (in^4)

L = bulkhead thickness or depth

M = bending moment (ft·lb)

M_u = factored beam moment (ft·lb)

r = maximum pressure head (88 psi)

S = section modulus (in^3)

T = tensile bending force (lb)

V_c = concrete shear strength (lb)

V_s = rebar shear strength (lb)

v_c = concrete shear stress (psi)

W = bulkhead load (lb)

$_g$ = pressure gradient $\left(\frac{psi}{ft}\right)$

r = ratio of nonprestressed tensile reinforcement to concrete compressivearea = $\left(\frac{A_s}{b_w d}\right)$

strength reduction factors
0.90 flexure rebar tension
0.85 concrete shear
0.65 plain concrete flexure

g_w = water density (62.4PCF)
g_c = concrete density (151PCF)
g_r = rock density (168 PCF)
s = flexure stress (psi)

Strength reduction factors (Reinforced Concrete ACI 318-89, rev 1992, Sec 9.3.2.2 & Sec 9.3.2.3; Plain Concrete, ACI 318.1-89, rev 1992, Sec 6.2.2)

Load factors (ACI 318R-92, Sec 9.2.2, 9.2.3, 9.2.5)

Static fluid load factor (F) = 1.4
Factor for fluid load under earthquake acceleration (F) = 1.05
Earthquake accelerated load factor (E) = 1.40

Hydraulic pressure gradient:

Low pressure grouting of concrete-rock contact but not rock, gradient allowable = 41 psi/ft (Garrett & Campbell-Pitt, 1958, Chekan, 1985, p11), with factor of safety of 4

Old Timer's Adit bulkheads, maximum pressure head

$$p = \frac{Hg_w}{144} = \frac{202(62.4)}{144} = 88 \text{ psi}$$

Required bulkhead thickness $L = \frac{p}{41} = \frac{88}{41} = 2.15 \text{ ft}$

Pressure gradient with L = 5 ft $r_g = \frac{p}{L} = \frac{88}{5} = 18 \frac{\text{psi}}{\text{ft}}$ $FS = \frac{41}{18} = 2.28$

Concrete shear, Old Timer's Adit bulkhead perimeter, 15 x 16 ft:

$f'_s = 2\sqrt{f'_c} = 2\sqrt{3000} = 110 \text{ psi}$ (ACI 318R-89, rev 1992, Sec 11.3.1.1)

$$L = \frac{{}_r\ hl}{\left[2(h+{}_l)f'_s\right]} = \frac{(1.4)88(7)5}{\left[2(7+5)110\right]} = 1.63 \text{ ft}$$

W = fp hl = (1.4)88a7f5a144f = 621000 lb

$$v_c = \frac{W}{\left[2(h+{}_l)\right]L(144)} = \frac{621000}{\left[2(7+5)\right]5(144)} = 35.9 \text{ psi}$$

$$FS = \frac{110}{35.9} = 3.06$$

Rock shear, Old Timer's Adit bulkhead perimeter, 7 x 5ft:

$f'_r \geq 210$ psi (minimum cohesion from physical test results)

$$\mathbf{L} = \frac{\phi p h \ell}{\left[2(h+\ell)f'_r\right]} = \frac{(1.4)88(7)5}{\left[2(7\ +\ 5)210\right]} = \mathbf{<0.86\ ft}$$

$$W = \phi p h \ell = (1.4)88(7)5(144) = 621000\ \text{lb}$$

$$v_s = \frac{W}{\left[2(h+\ell)\right]L(144)} = \frac{621000}{\left[2(7+5)\right]5(144)} = 35.9\ \text{psi}$$

$$\mathbf{FS} \geq \frac{210}{35.9} \geq \mathbf{5.85}$$

Plain concrete deep-beam bending stress design, Old Timer's Adit bulkhead perimeter, 15 x 16 ft:
(ACI 318R-89, rev 1992, Sec 9.9.2.5, 18.4.1(b), & ACI 318.1-89, rev 1992, Sec 6.2.2)

$$\omega = U = \phi p(144) = 1.4(88)144 = 17700\ \frac{\text{lb}}{\text{ft}}$$

$$M_n = \frac{\omega h^2}{8} = \frac{17700\left(15^2\right)}{8} = 498000\ \text{ft}\cdot\text{lb}$$

$$M_u = \frac{M_n}{0.65} = \frac{498000}{0.65} = 766000\ \text{ft}\cdot\text{lb}$$

$$S = \frac{I}{c} = \frac{\frac{bL^3}{12}}{\frac{L}{2}} = \frac{\frac{1\left(L^3\right)\left(12^3\right)}{12}}{\frac{L(12)}{2}} = \frac{144L^2}{6}$$

$$f'_{ct} = 3\sqrt{f'_c} = 3\sqrt{3000} = 164\ \text{psi (ACI 318-89, rev 1992, Sec 18.4.1(b))}$$

$$f'_{ct} = 5\phi\sqrt{f'_c} = 5(0.65)\sqrt{3000} = 178\ \text{psi (ACI 318.1-89, rev 1992, Sec 6.2.1(a) \& 6.2.2)}$$

$$f'_{ct} = 164 = \sigma = \frac{M_u c}{I} = \frac{M_u}{S} = \frac{766000}{\frac{144L^2}{6}} = \frac{31900}{L^2}$$

$$L = \sqrt{\frac{31900}{164}} = \mathbf{14.0\ ft}$$

∴ 5-ft long bulkhead must be reinforced

Critical section shear strength for Old Timer's Adit, 5-ft thick beam bulkhead in 5-ft wide drift

Deep beam defined as $\frac{\ell}{d} < 5$ (ACI 318-89 rev 1992, Sec 11.8.1) $\frac{\ell}{d} = \frac{5}{4.75} = 1.05 < 5.$ ∴ bulkhead is a deep beam for design

Detailed shear strength at critical section (ACI 318-89, rev 1992, Sec 11.8.7)

$$M_n = \left(\frac{\omega\ell}{2}\right)(0.15\ell) - \omega(0.15\ell)\frac{(0.15\ell)}{2} = 0.06375\omega\ell^2$$

$$M_n = 0.06375\left[17700(5^2)\right] = 28200 \text{ ft}\cdot\text{lb}$$

$$M_u = \frac{M_n}{0.9} = \frac{28200}{0.9} = 31300 \text{ ft}\cdot\text{lb}$$

$$V_n = \frac{\omega\ell}{2} - \left(\frac{\omega\ell}{2}\right)\left(\frac{0.15\ell}{0.5\ell}\right) = 0.35\omega\ell = 0.35(17700)5 = 31000 \text{ lb}$$

$$V_u = \frac{V_n}{0.85} = \frac{31000}{0.85} = 36500 \text{ lb}$$

$$K = 3.5 - 2.5\frac{M_u}{V_u d} = 3.5 - 2.5\frac{31300}{36500\left(\frac{57}{12}\right)} = 3.5 - 0.45 = 3.05$$

K cannot exceed 2.5 ∴ K = 2.5

$$\rho = \frac{A_s}{b_w d} = \frac{0.44}{12(57)} = 0.00064$$ Assumes only temperature and shrinkage rebar required

$$V_c = K\left[1.9\sqrt{f_c'} + 2500\rho\frac{V_u d}{M_u}\right]b_w d$$

$$V_c = 2.5\left[1.9\sqrt{3000} + 2500(0.00064)\frac{36500(4.75)}{15700}\right]12(57)$$

$$V_c = 2.5[104.1 + 17.7]684 = 2.5[121.8]684 = 208000 \text{ lb}$$

$$\mathbf{FS} = \frac{V_c}{V_u} = \frac{208000}{36500} = \mathbf{5.70}$$

Earthquake bulkhead design; Load factors (ACI 318 rev 1992, Sec 9.2.2, 9.2.3, 9.2.5) Factor for fluid load under earthquake acceleration (F) = 1.05; Load factor for earthquake accelerated mass (E) = 1.40

U = 1.05F + 1.40E

Factored fluid load = 1.05F =1.05(88)144 = 13300 lb/ft

Mass accelerated by earthquake (E)

$$E_m = \frac{S_\ell \gamma_w h\ell + Lh\ell\gamma_c}{32.2} = \frac{[400(62.4)7(5)+5(7)5(151)]}{32.2}$$

$$E_m = \frac{[874000 + 26400]}{32.2} = \frac{[900000]}{32.2} = 28000 \frac{\text{lb}\cdot\text{sec}^2}{\text{ft}} H_r$$

$$\Sigma E_m = E_m\alpha = 28000(0.016) = 448 \text{ lb}$$

$$E = \frac{\Sigma E_m}{\ell} = \frac{448}{7} = 64 \frac{\text{lb}}{\text{ft}}$$

Total load under earthquake acceleration

$$\rho = \frac{H\gamma_w}{144} = \frac{202(62.4)}{144} = 88 \text{ psi}$$

$$F = \rho b_w(12) = 88(12)12 = 12700 \frac{\text{lb}}{\text{ft}}$$

$$U_\alpha = 1.05F + 1.40E = 1.05(12700) + 1.40(64) = 13300 + 90 = 13400 \frac{\text{lb}}{\text{ft}}$$

Earthquake nominal beam bending moment

$$M_{n\alpha} = \frac{U_\alpha h^2}{8} = \frac{13400(7^2)}{8} = 82100 \text{ ft}\cdot\text{lb}$$

$$M_{u\alpha} = \frac{M_n}{} M_{u\alpha} = \frac{M_{n\alpha}}{0.9} = \frac{82100}{0.9} = 91200 \text{ ft}\cdot\text{lb } (1090000 \text{ in}\cdot\text{lb})$$

$$MM_{u\alpha} = M_{u\alpha} = M_{n\alpha} = \frac{U_\alpha h^2}{8} = \frac{13400(7^2)}{8} = 82100 \text{ ft}$$

Steel area required for earthquake loading

$$58800A_s^2 - 3420000A_s + 1090000 = 0\,58800A\,58800A_s^2 - 3420000A_s + 1090000 = 0$$

$A_s = 0.32 \frac{\text{in}^2}{\text{ft}}$ **steel area required to withstand design earthquake loading**

#6 bars on 12-in c-c required for temperature and shrinkage protection of concrete provide

$$\mathbf{0.44}\ \frac{\text{in}^2}{\text{ft}}\quad \textbf{steel area}$$

Check for adequacy

$$\text{Allowable}\ \ M_{u\alpha} = -58000A_s^2 + 3420000A_s = 1490000\ \text{in}\cdot\text{lb}$$

$$\text{Design}\ \ M_{u\alpha} = 1090000\ \text{in}\cdot\text{lb}$$

$$\mathbf{FS} = \frac{1490000}{1090000} = \mathbf{1.37}$$

Overburden depth (H_r) required to prevent hydrofracturing of adjacent rock (Einarson & Abel, 1990, p 71-15)

$$\mathbf{H_r} = \frac{72B_p}{\gamma_r} = \frac{72(88)}{168} = \mathbf{38\ ft}$$

John F. Abel, Jr.
Colorado P.E. 5642

Check for adequacy

$$\text{Allowable } M_{u\alpha} = -58800A_s^2 + 3420000A_s = 6700000 \text{ in} \cdot \text{lb}$$

$$\text{Design } M_{u\alpha} = 5110000 \text{ in} \cdot \text{lb Design } M_{u\alpha} = 5110000 \text{ in} \cdot \text{lb}$$

$$\mathbf{FS} = \frac{6700000}{5110000} = \mathbf{1.31}$$

Overburden depth (H_r) required to prevent hydrofracturing of adjacent rock (Einarson & Abel, 1990, p 71-15)

$$\mathbf{H_r} = \frac{72B_p}{\gamma_r} = \frac{72(88)}{168} = \mathbf{38\ ft}$$

John F. Abel, Jr.
Colorado P.E. 5642

APPENDIX J
HYDROLOGIC MONITORING PLAN

HYDROLOGIC MONITORING PLAN

For

Lamefoot Mine Site

Republic, Washington

TABLE OF CONTENTS

Section Page

1.0 INTRODUCTION 1
2.0 MINE WATER 3

  2.1 UNDERGROUND SEEPAGE 3
  2.2 MINE PORTAL/INFILTRATION POND 4

3.0 SURFACE WATER 5
4.0 GROUNDWATER 7

  4.1 SHALLOW MONITORING WELLS 7
  4.2 DEEP MONITORING WELLS 8
  4.3 WATER SUPPLY WELLS 11
  4.4 GROUNDWATER LEVEL MONITORING AND SAMPLING 11

5.0 POST-CLOSURE MONITORING 13
6.0 RECORDKEEPING AND REPORTING 14

LIST OF TABLES

TABLE J-1 LAMEFOOT MINE HYDROLOGIC MONITORING PROGRAM
TABLE J-2 PARAMETERS AND ANALYTICAL METHODS FOR WATER (METALS)
TABLE J-3 PARAMETERS AND ANALYTICAL METHODS FOR WATER (ANIONS AND OTHERS)
TABLE J-4 LAMEFOOT MINE HYPOTHETICAL POST-CLOSURE HYDROLOGIC MONITORING PROGRAM

LIST OF FIGURES

FIGURE J-1 HYDROLOGIC MONITORING LOCATIONS
FIGURE J-2 WATER WELL INVENTORY MAP
FIGURE J-3 WATER QUALITY MONITORING BY YEAR

STANDARD OPERATING PROCEDURES

SOP NO. 1 EQUIPMENT DECONTAMINATION
SOP NO. 2 GROUNDWATER MONITORING WELL AND PIEZOMETER INSTALLATION
SOP NO. 3 FIELD PACKER TESTING
SOP NO. 4 AQUIFER PUMPING TESTS
SOP NO. 5 WATER LEVEL MEASUREMENT
SOP NO. 6 MONITORING WELL SAMPLING
SOP NO. 7 SAMPLE HANDLING, DOCUMENTATION AND ANALYSIS

1.0

INTRODUCTION

Washington Department of Ecology (DOE) Waste Water Discharge Permit No. 8033 addresses monitoring at the Lamefoot Mine infiltration pond, Key Mill and Tailings Facility, and three other Key Project Mines. DOE Permit No. 8033 is attached as Appendix E. The Hydrologic Monitoring Plan (HMP) for the Lamefoot Mine incorporates the relevant monitoring and reporting requirements of DOE Waste Water Discharge Permit No. 8033 (i.e., Sections S2, S3, S4, and S7), as well as additional monitoring that has been required or recommended as mitigation measures by the Bureau of Land Management (BLM). These requirements were established through the processes of permitting the Lamefoot exploration activities and preparation of the DEISS.

The Hydrologic Monitoring Plan addresses the monitoring and reporting activities throughout the operational period of the Lamefoot Mine. In particular, the HMP addresses the monitoring locations and sampling frequencies for mine water discharge, surface water (i.e., springs and wetlands), and groundwater at the mine site, and recordkeeping and reporting requirements. The HMP includes the following items:

1. Map of hydrologic monitoring locations
2. Construction specifications for proposed wells
3. Aquifer testing procedures
4. Monitoring frequency
5. Sample collection procedures
6. Chemical parameters and analytical methods

Standard operating procedures (SOPs) for equipment decontamination, drilling and well installation, aquifer testing, sampling, and documentation are attached as part of the HMP and referenced throughout the HMP as appropriate. The SOPs were established to guide field personnel during the implementation of the monitoring program and assure the collection of consistent, reproducible, and defendable data.

The Hydrologic Monitoring Plan for the period of active mining at the Lamefoot Mine is summarized in Table J-1. Sample locations are indicated on Figure J-1. Tables J-2 and J-3 present the analytes and methods of laboratory analyses. The complete list of parameters will be analyzed for laboratory samples collected during the first year of monitoring, and for one sample annually thereafter. The "short list" of parameters indicated on Tables J-2 and J-3 will be analyzed for the remainder of intervening samples.

A hypothetical Post-Closure Monitoring Program is summarized on Table J-4. Laboratory sample analysis parameters would include the same complete list and short list of parameters indicated on Tables J-2 and J-3. The water quality monitoring program by year for the life of the mine and post-closure is depicted graphically on Figure J-3.

2.0
MINE WATER

Mine water in this Plan includes (a) water discharging as underground seepage into the mine, and (b) discharge of this water at the surface from the portal into the infiltration pond. Monitoring will consist of quantifying mine water discharge, and measuring field and laboratory water quality. Table J-1 presents the monitoring frequency for the discharge and water quality measurements.

2.1 UNDERGROUND SEEPAGE

The amount of underground seepage into the mine will be monitored on a bimonthly basis. Estimates of flow from individual seeps will be summed to provide an estimate of total inflow to the mine. The monitoring of mine inflows will be used to identify any substantial (i.e., > 1 gpm) sources of inflow to be grouted for mine water control. If conditions permit, seepage will be estimated using a container of known volume to collect the discharge and recording the amount of time it takes to fill the container. If conditions prevent this method from being employed, a visual estimate will be made using best professional judgement.

Water quality monitoring will be conducted to identify whether any specific locations of degraded water quality inflows occur within the mine. Quarterly field water quality measurements from discrete inflow points of greater than 1.0 gpm will be recorded. Characterization of field water quality will consist of collecting a representative sample of the seepage water and taking pH, specific conductivity, and temperature measurements using a portable water quality kit. The procedure for taking field water quality measurements is described in SOP No. 6.

Monitoring data from underground seeps will be recorded on a current map of the underground workings and also in tabular format. The map will also show the locations and dates of grouting so that the mine inflow measurements can be used to evaluate the effectiveness of the grouting program.

2.2 MINE PORTAL/INFILTRATION POND

Mine water that is not used within the mine for drilling is discharged from the portal to the infiltration pond. This discharge is regulated by DOE Permit No. 8033. The portal discharge is intermittent, and is currently monitored using a totalizing flow meter. Meter readings and field water quality parameters will be recorded three times each week in accordance with the requirements of DOE Permit No. 8033 (Section S2). During the periodic monitoring of the infiltration pond, the observed water level will be recorded and amount of accumulated sediment will be noted. Sediment will be removed as necessary to maintain the volume and infiltration capacity of the pond, as specified in DOE Permit No. 8033 (Section S2).

Field water quality measurements of water discharging from the portal into the infiltration pond will be collected three times a week (conditions permitting) in accordance with the requirements of DOE Permit No. 8033 (Section S2). The purpose of this frequent sampling is to monitor general water quality changes in the mine water discharge which could indicate a permit exceedance. Field water quality measurements will be collected at the point of discharge to the pond if any flow is occurring at the time of measurement. Otherwise, the field water quality measurements will be obtained by collecting a sample of the standing water in the settling or infiltration ponds.

In addition to the field water quality monitoring, monthly samples will be collected for laboratory water quality analyses. The purpose of this sampling is to monitor changes in mine water chemistry which may be naturally occurring or induced from mining operations and to maintain permit discharge compliance. Tables J-2 and J-3 present the specific analytes and analytical methods to be performed. Sample containers will be supplied by the contracted analytical laboratory. Procedures for decontaminating sampling equipment and collecting laboratory water quality samples are described in SOP No. 1 and SOP No. 6, respectively.

3.0
SURFACE WATER

Surface water in this Plan includes (a) water discharging from springs, and (b) water contained within the various wetland areas at the mine site. Surface water monitoring will consist of quantifying discharge and water levels, as well as measuring field and laboratory water quality. These measurements will be used to document natural variations in surface water quantity and quality, and to identify potential impacts of mining that may indicate the need for mitigation. Table J-1 presents the monitoring frequency for the surface water discharge and water quality measurements.

Measurements of flow rate will be made monthly at two springs. One of the springs is located within the Wolfe Camp drainage an elevation of 2,605 feet (Spring 2605), while the other spring is located northeast of the permit boundary at an elevation of 3,035 feet (Spring 3035). A V-notch weir is currently installed below Spring 2605 and will continue to be used to monitor discharge from the spring. Based on a preliminary reconnaissance of Spring 3035, discharge is reported to be quite low and will be estimated visually using best professional judgement. Procedures for measuring flow at Spring 2605 with the V-notch weir are described in SOP No. 5.

Visual observations of water conditions will be made monthly at four wetland areas. The wetland areas are located within the permit boundary at elevations of 2,845 feet (Wetland 2845), 2,545 feet (Wetland 2545), 2,540 feet (Wetland 2540), and 2,495 feet (Wetland 2495). During each monthly visit, conditions such as standing water, wet or boggy ground, or visible discharge will be noted, if observed. If discharge from the wetland is observed, a visual estimate of the flow will be made and recorded. If standing water is present, water elevation from a surveyed staff gauge will be measured and recorded.

During the monthly visits to the springs and wetlands, field water quality measurements will be performed at each location if there is a measurable water level. If water is not present it will be noted. The purpose of this sampling is to establish a characterization of general water quality at these locations which can be used as a possible indicator for mining impacts. Characterization of field water quality will consist of collecting a representative sample of water from the spring or wetland and measuring the pH, specific conductivity, and temperature using a portable water quality kit. The procedure for taking field water quality measurements is described in SOP No. 6.

Samples for laboratory water quality analyses will be collected every quarter from Spring 2605, and will be coincident with quarterly sampling of groundwater monitoring wells. The purpose of this sampling is to monitor possible seasonal variations in water chemistry (which are naturally occurring), or variations which may have been induced from mining operations. In addition, laboratory water quality analyses will be collected from Spring 3035, if conditions permit. Two samples will be collected (an initial sample and a confirmation sample) from this spring during separate sampling events. The purpose of this sampling is to document baseline water chemistry in the off-site spring so that potential future mining impacts could be assessed, if necessary, at this location.

Tables J-2 and J-3 present the specific laboratory water quality analytes and analytical methods to be performed. Procedures for decontaminating sampling equipment and collecting laboratory water quality samples are described in SOP No. 1 and SOP No. 6, respectively.

4.0
GROUNDWATER

Under this Plan, groundwater conditions will be monitored in two existing monitoring wells, seven new monitoring wells, and several existing private water wells (Figure J-1). This network of monitoring wells is designed to detect possible changes in water quality or water level (a) during operation and post-closure of the mine, (b) from seepage through ore/waste rock stockpiles, and (c) from seepage below infiltration pond. The following tasks will be performed as part of the groundwater monitoring program:

- Drilling, installation, and development of four new shallow monitoring wells
- Drilling, selective hydraulic testing, installation, and development of three new bedrock wells
- Groundwater level monitoring and sampling of the existing and new monitoring wells, and selected private wells.

Each of these tasks are described below.

4.1 SHALLOW MONITORING WELLS

Four new monitoring wells (LF-5, LF-7, LF-8, and LF-9) will be installed within the shallow unconsolidated sediments. The approximate locations of these wells are shown on Figure J-1. The purpose of these wells is to document natural variations in groundwater flow directions and water quality within the unconsolidated sediments, and to identify potential impacts from mining that may indicate the need for mitigation.

These four wells are anticipated to be less than 50 feet in depth and will be drilled and installed using air rotary or dual tube percussion drilling methods. An exception is well LF-9, which was installed as a driven well point because of access constraints at this location. During drilling, borehole stratigraphy will be logged by examination of the sample cuttings collected at regular intervals or at changes in lithology. No soil or rock samples will be collected for chemical analysis.

Boreholes for these holes will be advanced to a depth of 20 feet below the water table or to the contact with bedrock. Once the targeted depth is reached, well construction will proceed with the placement of well construction materials through the inside of the drill casing. Each well will be completed using 2-inch diameter, Schedule 40, threaded PVC casing and well screen. Well screens will have factory-machined slots of 0.02 inches and will be a maximum of 20 feet in length. Well screen bottoms will be fitted with a 2-foot sump or cap.

Following placement of the well casing and screen, a gravel pack consisting of #10-20 silica sand will be placed in the well annulus opposite the well screen and extending 2 to 3 feet above the top of the screen. A bentonite pellet seal will be placed above the gravel pack, followed by the placement of a cement/bentonite grout seal above the bentonite seal. The cement/bentonite grout seal will extend to ground surface. Each well will be completed by installing a locking steel protective casing around the well casing and placing a PVC well cap over the well casing.

Following well construction, the elevation of the top of the well casing and ground surface will be surveyed to within 0.01 feet, and tied into a documentable bench mark. The wells will be developed appropriately before sampling is to commence. Procedures for drilling, installing, and developing the wells are described in SOP No. 2.

4.2 DEEP MONITORING WELLS

Three new monitoring wells (LF-3, LF-4, and LF-6) will be installed within the deep bedrock beneath the mine site. The approximate locations of these wells are also shown on Figure J-1. The deep wells will be used to further characterize the permeability, flow pathways, and hydraulic connection with shallow groundwater for baseline characterization, in addition to their use for monitoring water quality within the bedrock.

The depths of these wells are expected to range from about 175 to 650 feet below ground surface. The wells will be drilled and installed using the air rotary drilling method. During drilling, borehole stratigraphy will be logged by examination of the sample cuttings collected at regular intervals, at changes in lithology, or where fracturing of the bedrock mass becomes noticeable. No soil or rock samples will be collected for chemical analysis.

Boreholes for these holes will be advanced through the unconsolidated sediments to the contact with bedrock. Once the bedrock contact is reached, the borehole will be advanced into the bedrock mass a sufficient distance in preparation of setting a surface casing which will isolate the shallow groundwater in the unconsolidated sediments from groundwater in the deeper bedrock. An 8-inch diameter steel surface casing will be installed and grouted in place. Before proceeding with further drilling, the grout will be allowed to set a minimum of 24 hours to stabilize the surface casing. After the grout has set, drilling will proceed through the inside of the surface casing using a nominal 6-inch diameter bit until the targeted well depth is reached.

Once the targeted depth is reached, the borehole will be cleared of cuttings and preparations for packer testing of the bedrock mass will begin. Each borehole will be packer tested on 20-foot intervals to evaluate the hydraulic characteristics of discrete zones of the bedrock. Of most interest are zones of noticeable fracturing and faulting, because these zones are most likely to provide the pathways for the migration of groundwater through the bedrock. Packer testing consists of isolating a portion of the bedrock and injecting water into the formation under pressure. The injection rate is used to estimated the permeability of the isolated portion of the bedrock. Packer testing will be performed according the procedures outlined in SOP No. 3.

Following completion of the packer tests in boreholes LF-4 and LF-6, constant rate pump tests will be performed. These tests will be performed to evaluate the average hydraulic characteristics of the bedrock aquifer; and to qualitatively evaluate the possible hydraulic interconnection between groundwater in the overlying unconsolidated sediments, and groundwater in the bedrock. These tests will be performed for a period of 8 to 12 hours. During the tests, water will be pumped at a constant rate, and water levels will be monitored in the pumping well and in nearby monitoring wells for drawdown response. At the completion of the test, water levels in these same wells will be monitored for complete recovery to static levels. The pumping rate to be used during each test will be based on preliminary observations of well development rates and a preliminary assessment of the packer test results. Procedures for conducting the pump tests are described in SOP No. 4.

After completing the pump tests, work will begin on completing monitoring wells in the three boreholes. Based on lithologic and hydraulic testing information, an interval will be targeted near the bottom of the borehole for well completion. If necessary, the borehole will be backfilled to the target interval before construction begins. Well construction will proceed with the emplacement of well materials through the inside of the drill casing.

Each well will be completed using 2-inch diameter, Schedule 80, threaded PVC casing and well screen. Well screens will have factory-machined slots of 0.02 inches and will be a maximum of 20 feet in length. Well screen bottoms will be fitted with a 2-foot sump or cap.

Following placement of the well casing and screen, a gravel pack consisting of #8-12 silica sand will be placed in the well annulus opposite the well screen and extending 2 to 3 feet above the top of the screen. A bentonite pellet seal will be placed above the gravel pack, followed by the placement of a cement/bentonite grout seal above the bentonite seal. The cement/bentonite grout seal will extend to ground surface, and will be placed in two separate lifts. The first lift of grout will be placed the first 100 feet above the bentonite seal and will be allowed to set 24 hours before placing the remaining grout in the second lift. Each well will be completed by installing a locking steel protective casing around the well casing and placing a PVC well cap over the 2-inch well casing.

Following well construction, the elevation of the top of the well casing and ground surface will be surveyed to within 0.01 feet, and tied into a documentable bench mark. The wells will be developed appropriately before sampling is to commence. Procedures for drilling, installing, and developing the wells are described in SOP No. 2.

4.3 WATER SUPPLY WELLS

Several water supply wells will be monitored annually as part of the monitoring program. This monitoring will supplement the data obtained from the monitoring wells and will be used to identify any potential impacts to the supply wells that may require mitigation.

The water supply wells to be monitored include the Heckley, Echo Bay and State Park wells which are shown on Figure J-2 in relation to the Project Area and other wells on file with Washington DOE.

4.4 GROUNDWATER LEVEL MONITORING AND SAMPLING

Groundwater levels will be monitored on a monthly basis in the existing and new monitoring wells (Table J-1). The purpose of this monitoring is to evaluate groundwater flow directions in the unconsolidated sediments and the bedrock, and to evaluate water level fluctuations which may be attributable to the seasons or mining operations. Water levels will also be monitored in private water wells prior to sampling. The procedure and necessary equipment required for water level monitoring is described in SOP No. 5.

The two existing shallow monitoring wells and six new monitoring wells will be monitored quarterly for field and laboratory water quality parameters. This network of monitoring wells is designed to detect possible changes in water quality during operation and post closure of the mine, ore/waste rock stockpiles, and infiltration pond. Table J-1 presents the monitoring frequency for the groundwater quality measurements.

Additionally, several private water supply wells will be monitored annually as part of the program. This monitoring will supplement the data obtained from the existing and new monitoring wells and will be used to identify any potential impacts to the supply wells that may require mitigation.

Tables J-2 and J-3 present the specific laboratory water quality analytes and analytical methods to be performed. Sample containers will be supplied by the contracted analytical laboratory. Procedures for decontaminating sampling equipment and collecting laboratory water quality samples are described in SOP No. 1 and SOP No. 6, respectively.

5.0
POST-CLOSURE MONITORING

In accordance with DOE Permit No. 8033, monitoring of groundwater, surface water, and/or mine water will be required following closure of the mine operations. Establishment of a monitoring plan for this period is premature without sufficient historical knowledge of the geochemical trends of these waters. The number and location of monitoring points and the types of analytes that will be required is dependent upon this information and upon Washington DOE requirements at the time of closure. The historical sampling data will be able to indicate whether some monitoring points no longer require monitoring, or whether additional monitoring points may be required.

It is assumed that the proposed monitoring network would be sufficient for post-closure monitoring. A hypothetical monitoring frequency is presented in Table J-4. After 10 years of post-closure monitoring, a selected number of sample locations would be monitored once every two years to detect long-term geochemical changes during years 13 through 30 after mine closure. This hypothetical case has been presented only for purposes of estimating the amount needed for post-closure bonding.

A post-closure monitoring plan will be prepared prior to mine closure, which will take into consideration the historical monitoring data. The post-closure monitoring plan will outline the most appropriate monitoring locations, monitoring frequencies, and analytes for this period, along with appropriate reporting procedures. Monitoring water quality in the backfilled mine will be accomplished by sampling from two wells that would be installed into the backfilled mine workings after closure.

Laboratory analyses of the samples indicated on Table J-4 would be analyzed for the parameters listed on Tables J-2 and J-3. The complete list of parameters would be analyzed for the first year after mine closure. During subsequent years, the complete list will be analyzed only once per year (or every two years for biannual samples) with intervening quarterly samples analyzed for the "short list" of parameters as indicated on the Tables J-2 and J-3. The Hydrologic Monitoring Program by year for the life of mine and post-closure is presented graphically on Figure J-3.

6.0
RECORDKEEPING AND REPORTING

Recordkeeping and reporting will be in accordance with the requirements of DOE Permit No. 8033 (Section S3), and documentation procedures presented in the SOPs which are attached. At a minimum, all records of monitoring activities and results shall be retained for the operational life of the mine, should any further actions become necessary. Records and results will be kept in the files at the mine site.

Monitoring results will be reported to the Washington DOE and the BLM or other requirements specified by BLM on a monthly basis. Reporting and filing will be in accordance with the requirements of DOE Permit No. 8033 (Sections S3 and S7). The report will present a description of the monitoring activities performed that month, a map showing the location of all sampling points, tabulated measurements, and the laboratory analytical results and will be summarized in an annual interpretive report. Annual report information will include, but will not be restricted to text, tables and/or maps describing:

- All Environmental Monitoring Data
- Summary of Mine Development Activities
- Summary of Surface Construction Activities
- Summary of Reclamation Activities
- Summary of Operation Schedule
- Production Records

In the case of post-closure monitoring, reporting frequency will coincide with the monitoring frequency, for example quarterly monitoring and reporting would be performed during the first 10 years after mine closure based upon the schedule presented in Table J-4.

TABLE J-1

LAMEFOOT MINE
HYDROLOGIC MONITORING PROGRAM DURING OPERATIONS

Sample Location	Flow	Water Level	Field Water Quality	Laboratory Water Quality[1]	Comments
Mine Water					
Underground Seeps	M		Q		Summed to estimate inflow
Portal	3/W				Totalizing meter installed
Infiltration Pond		3/W	3/W	M (O&G)	Weekly visual inspection
Surface Water					
Spring at 2605'	M		M	Q	V-notch weir installed
Spring at 3035'[2]	M		M	2B	N.E. of permit area
Wetland at 2495'		M	M		
Wetland at 2540'		M	M		
Wetland at 2545'		M	M		
Wetland at 2845'		M	M		
Groundwater Monitoring Wells					
LF-1 (shallow)		M	Q	Q	
LF-2 (shallow)		M	Q	Q	
LF-3 (deep)		M	Q	Q	N. of mine
LF-4 (deep)		M	Q	Q	W. of mine
LF-5 (shallow)		M	Q	Q	N. of inf. pond
LF-6 (deep)		M	Q	Q	W. of inf. pond
LF-7 (shallow)		M	Q	Q	W. of inf. pond
LF-8 (shallow)		M	Q	Q	W. of stockpiles
LF-9 (shallow)		M	Q	Q	S. of stockpiles
Private Water Wells					
Mine Supply Well (Echo Bay)		A	A	A	
Heckley (Echo Bay)		A	A	A	N.W. of mine
State Park[2]		A	A	A	Near Curlew Lake

Explanation of Symbols:
W = weekly
M = monthly
Q = quarterly
A = annually
2B = Two baseline samples to be collected during first year of monitoring

Water Quality Parameters:
Field = pH, conductivity, temperature

Laboratory = See parameter list on Tables J-2 and J-3.
(O&G) = Oil and grease analysis in addition to parameter list

[1] Complete list of laboratory parameters to be analyzed for the first year of monitoring. During subsequent years the complete list will be analyzed once per year and the short list of parameters will be analyzed for the remaining samples.

[2] These will be monitored if access can be arranged.

TABLE J-2

PARAMETERS AND ANALYTICAL METHODS FOR WATER[1] (METALS)

Parameter	Short List[4]	Complete List[3]	Analytical Method[2] Recommended Method
Aluminum (Al)		X	EPA Method 200.7
Antimony (Sb)		X	EPA Method 200.7
Arsenic (As)	X	X	EPA Method 206.2
Barium (Ba)		X	EPA Method 200.7
Beryllium (Be)		X	EPA Method 200.7
Cadmium (Cd)		X	EPA Method 200.7
Calcium (Ca)		X	EPA Method 200.7
Chromium (Cr), Total		X	EPA Method 200.7
Copper (Cu)	X	X	EPA Method 200.7
Iron (Fe)	X	X	EPA Method 200.7
Lead (Pb)	X	X	EPA Method 239.2
Magnesium (Mg)		X	EPA Method 200.7
Manganese (Mn)	X	X	EPA Method 200.7
Mercury (Hg)	X	X	EPA Method 245.1 or 245.2
Molybdenum (Mo)	X	X	EPA Method 200.7
Nickel (Ni)		X	EPA Method 200.7
Potassium (K)		X	EPA Method 200.7
Selenium (Se)	X	X	EPA Method 270.2
Silicon		X	EPA Method 200.7
Silver (Ag)		X	EPA Method 200.7
Sodium (Na)		X	EPA Method 200.7
Thallium (Tl)		X	EPA Method 279.2
Zinc (Zn)	X	X	EPA Method 200.7

[1] NOTE: Samples for metals analyses will be and analyzed for total recoverable metals. Selected samples may be field-filtered and analyzed for dissolved metals.

[2] EPA Methods are NPDES-approved, per 40 CFR 136 Tables: EPA 200 Methods from Methods for Chemical Analysis of Water and Wastes, EPA 600/4-79-020, March, 1979, Revised March 1983.

[3] Complete list of parameters to be analyzed during the first year of monitoring then annually thereafter.

[4] Short list of parameters to be analyzed for the remainder of intervening sampled intervals.

TABLE J-3

PARAMETERS AND ANALYTICAL METHODS FOR WATER (ANIONS AND OTHERS)

Parameter	Short List[3]	Complete List[2]	Analytical Method[1] Recommended Method
Alkalinity, Total, HCO_3/CO_3 and Hydroxide	X	X	EPA Method 310.1
Ammonia (NH_3) - as N	X	X	EPA Method 350.1
Chloride (Cl)		X	EPA Method 325.1
Orthophosphate		X	EPA Method 365.1, 365.2, or 365.3
Hardness		X	Calculated from Ca and Mg
Fluoride (F)		X	EPA Method 340.2
Nitrate (NO_3) - as N		X	EPA Method 300.0
Nitrate + Nitrite (NO_3 + NO_2) - as N	X	X	EPA Method 353.2
Specific Conductance	X	X	EPA Method 120.1
Sulfate (SO_4)	X	X	EPA Method 375.4
Total Suspended Solids (TSS)	X	X	EPA Method 160.2
Total Dissolved Solids (TDS)	X	X	EPA Method 160.1

pH?

[1] EPA Methods are NPDES-approved, per 40 CFR 136 Tables: EPA 100 and 300 Methods from Methods for Chemical Analysis of Water and Wastes, EPA 600/4-79-020, March, 1979, Revised March 1983.

[2] Complete list of parameters to be analyzed during the first year of monitoring then annually thereafter.

[3] Short list of parameters to be analyzed for the remainder of intervening sampled intervals.

TABLE J-4

LAMEFOOT MINE
HYPOTHETICAL POST-CLOSURE
HYDROLOGIC MONITORING PROGRAM

Sample Location	Flow	Water Level	Field Water Quality	Laboratory Water Quality[1]	Comments	Duration (Years)
Mine Water						
Behind Sealed Portal-approx. elev. 2610'		Q	Q	A (O&G)	300 ft. well required in backfilled workings.	10/30[2]
Deepest Workings - approx. elev. 2400'		Q	Q	A (O&G)	500 ft. well required in backfilled workings.	10/30[2]
Surface Water						
Spring at 2605'	Q		Q	Q	V-notch weir installed	10/30[2]
Wetland at 2495'		Q	Q			10
Wetland at 2540'		Q	Q			10
Wetland at 2545'		Q	Q			10
Wetland at 2845'		Q	Q			10
Groundwater Monitoring Wells						
Existing:						
LF-1 (shallow)		Q	Q	Q		10/30[2]
LF-2 (shallow)		Q	Q	Q		10/30[2]
LF-3 (deep)		Q	Q	Q	N. of mine	10
LF-4 (deep)		Q	Q	Q	W. of mine	10/30[2]
LF-5 (shallow)		Q	Q	Q	N. of inf. pond	10/30[2]
LF-6 (deep)		Q	Q	Q	W. of inf. pond	10/30[2]
LF-7 (shallow)		Q	Q	Q	W. of inf. pond	10/30[2]
LF-8 (shallow)		Q	Q	Q	W. of stockpiles	10
LF-9 (shallow)		Q	Q	Q	S. of stockpiles, if accessible	10
Private Water Wells						
Mine Supply Well (Echo Bay)		A	A	A		10/30[2]
Heckley (Echo Bay)		A	A	A	N.W. of mine	1
State Park		A	A	A	Near Curlew Lake	1

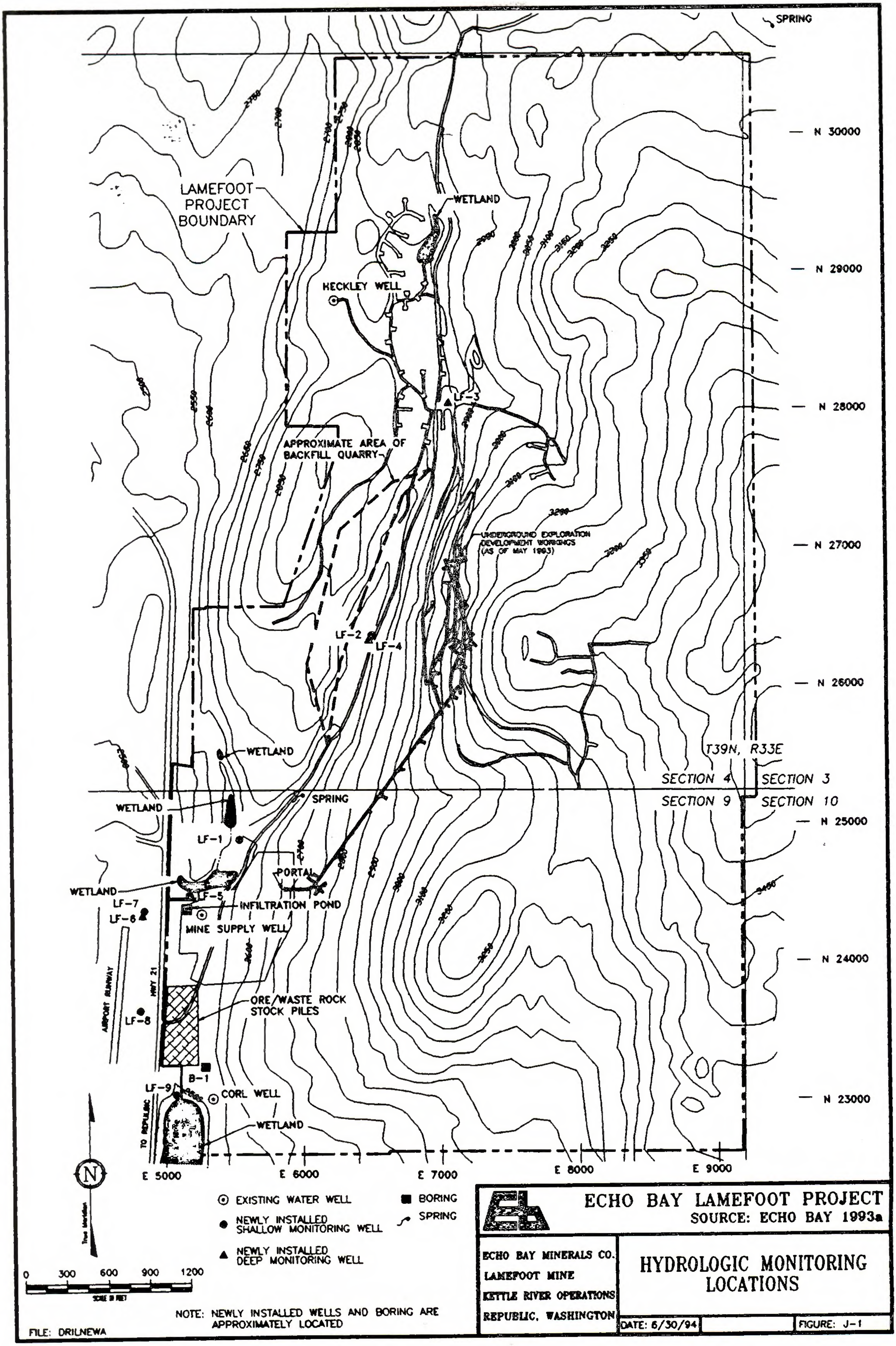
Explanation of Symbols:
Q = quarterly
A = annual

Water Quality Parameters:
Field = pH, conductivity, temperature
Laboratory = see parameter list on Tables J-2 and J-3
O&G = Oil and Grease in addition to list

---

[1] Complete list of laboratory parameters to be analyzed quarterly for all samples during the first year after closure, then once per year during years 2-10, and once per 2 years during years 12-30. The short list of parameters will be analyzed for quarterly samples during years 2-10.

[2] After initial 10-year monitoring, selected locations will be monitored once every two years from year 12 through year 30 after closure.



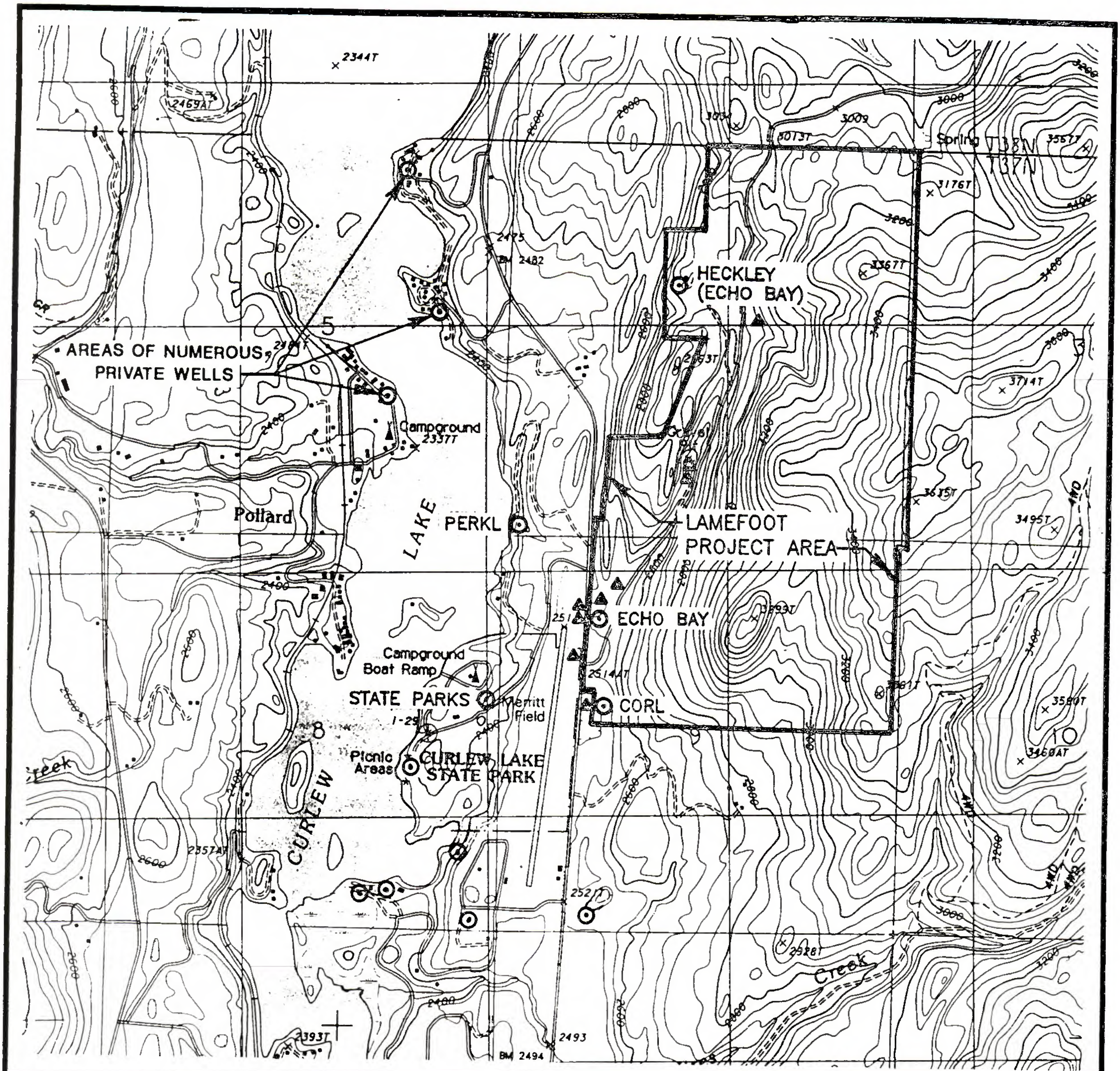
ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1993a

ECHO BAY MINERALS CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

HYDROLOGIC MONITORING LOCATIONS

DATE: 6/30/94

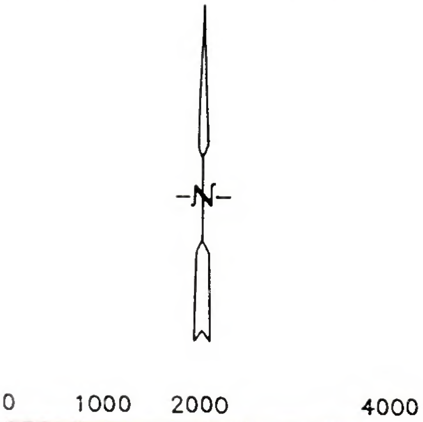
FIGURE: J-1



WELLS WITHIN 1/2 MILE OF PROJECT AREA

⊙ WATER WELL ON FILE WITH WASHINGTON D.O.E. LOCATION TO NEAREST 1/4 SECTION AS INDICATED ON WELL LOG.

▲ ECHO BAY MONITORING WELL



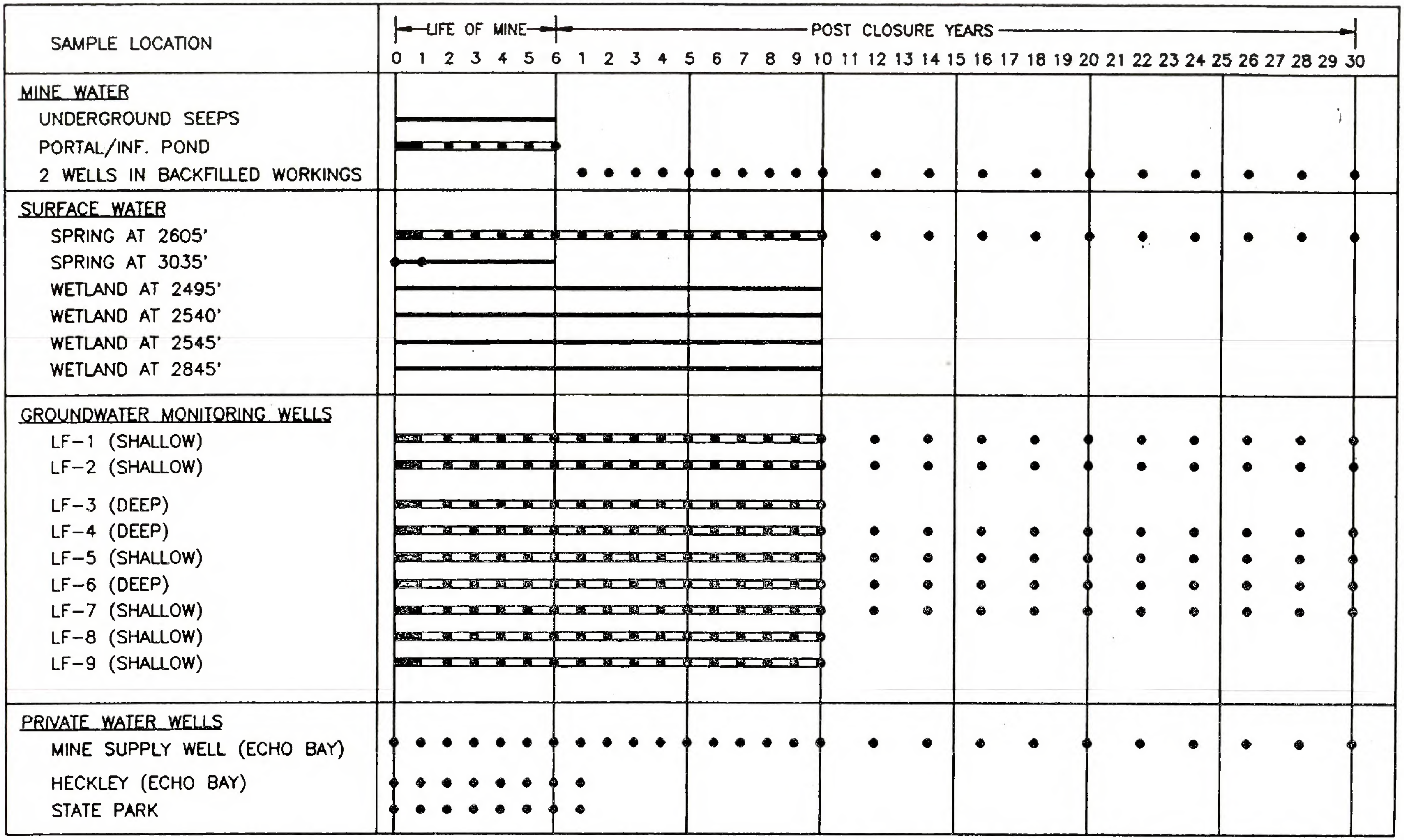
ECHO BAY LAMEFOOT PROJECT

SOURCE: ECHO BAY 1994

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

WATER WELL INVENTORY MAP

DATE: JUNE 1994 SCALE: 1" = 2000' FIG. NO. J-2



EXPLANATION

▬	QUARTERLY OR MORE FREQUENT MONITORING/LONG LIST
▭	QUARTERLY OR MORE FREQUENT MONITORING/SHORT LIST
—	QUARTERLY OR MORE FREQUENT MONITORING/FIELD PARAMETERS
●	ANNUAL MONITORING/LONG LIST

ECHO BAY LAMEFOOT PROJECT
SOURCE: ECHO BAY 1994

ECHO BAY MINERAL CO.
LAMEFOOT MINE
KETTLE RIVER OPERATIONS
REPUBLIC, WASHINGTON

LAMEFOOT MINE WATER QUALITY MONITORING BY YEAR

DATE: JUNE 1994 | SCALE: NA | FIG. NO. H-3

LAME-J3

APPENDIX J-1

SOP NO. 1 - EQUIPMENT DECONTAMINATION

1.0
TITLE PAGE

STANDARD OPERATING PROCEDURE NO. 1

EQUIPMENT DECONTAMINATION

LAMEFOOT MINE SITE

2.0
TABLE OF CONTENTS

Section | Page
1.0 TITLE PAGE 1
2.0 TABLE OF CONTENTS 2
3.0 INTRODUCTION 3
4.0 RESPONSIBILITIES AND QUALIFICATIONS 4
5.0 PROCEDURE 5
  5.1 EQUIPMENT LIST 5
  5.2 DECONTAMINATION 6
    5.2.1 Sampling Equipment 6
    5.2.2 Drilling and Heavy Equipment 7
  5.3 DOCUMENTATION 7

3.0

INTRODUCTION

The purpose of this document is to define the standard operating procedure for equipment decontamination for the Lamefoot Mine Site. This Standard Operating Procedure (SOP) serves as a supplement to the site-specific Hydrologic Monitoring Plan (HMP) and is intended to be used with the other SOPs.

The overall objective of multimedia sampling programs is to obtain samples which accurately depict the chemical, physical, and/or biological conditions at the sampling site. Extraneous contaminant materials can be brought to a sampling location and/or introduced into the medium of interest during the sampling program (e.g., by bailing or pumping of groundwater with equipment previously contaminated at another sampling site). Trace quantities of these materials can contaminate the sample and lead to false positive analytical results and, ultimately, to an incorrect assessment of the conditions associated with the site. Decontamination of sampling equipment (e.g., bailers, pumps, tubing, soil, and sediment sampling equipment) and field support equipment (e.g., drill rigs, vehicles) is required to ensure that sampling cross-contamination is prevented, and that on-site contaminants are not carried off site.

4.0

RESPONSIBILITIES AND QUALIFICATIONS

The Project Manager (Echo Bay's or consultant's) is responsible for assuring that equipment decontamination is conducted according to this procedure. The Project Manager will designate qualified project staff to complete this procedure.

The designated project staff are responsible for completing the equipment decontamination process according to this procedure. They report their progress, and any problems, to the Task Leader or Project Manager. Staff members are responsible for understanding the activities assigned to them and the quality assurance requirements associated with the activities. Problems related to equipment decontamination are also the responsibility of the Task Leader.

5.0
PROCEDURE

5.1 EQUIPMENT LIST

The following is a list of equipment that may be needed to perform decontamination:

- Brushes
- Wash tubs (minimum of 3) or
- 5-gallon buckets (minimum of 3)
- Scrapers
- Steam cleaner or high-pressure sprayer (portable)
- Large metal horse trough
- Disposal drums
- Sponges
- Paper towels
- Alconox detergent (or equivalent)
- Nitric acid (5% by volume) or hydrochloric acid (0.1N)
- Potable tap water
- Deionized organic-free water
- Garden type water sprayers
- Plastic sheeting
- Plastic trash bags
- Spray bottles
- Stainless steel bowls
- Appropriate health and safety equipment
- Field notebook

5.2 DECONTAMINATION

5.2.1 Sampling Equipment

The following steps will be used to decontaminate small sampling equipment, such as bailers, stainless steel trowels, stainless steel liners, plastic caps for liners, etc.:

- Personnel will dress in suitable safety equipment.
- Equipment showing gross contamination will be scraped off with a flat-bladed scraper.
- Equipment that will not be damaged by water will be placed in a wash tub containing Alconox (or low-sudsing detergent) along with potable water and scrubbed with a bristle brush or similar utensil. Equipment will be rinsed with clear potable water, in a second wash tub or bucket.
- When sampling for metals, the potable water rinse of the equipment will be followed by a 0.1N nitric acid (or 0.1N hydrochloric acid) rinse followed by a deionized water rinse.
- Equipment that may be damaged by water, such as a water quality kit, will be carefully wiped clean using a sponge and detergent water, and wiped with deionized water. Care will be taken to prevent any equipment damage.
- Detergent and rinse waters will be replaced between borings with fresh detergent and rinse waters.

Decontamination of sampling equipment will occur between each sample location. Following decontamination, sampling equipment will be placed in a clean area on clean plastic sheeting. If the equipment is not used immediately, it will be covered or wrapped in plastic sheeting to minimize potential airborne contamination.

5.2.2 Drilling and Heavy Equipment

Prior to moving onto each site, drilling and heavy equipment shall be decontaminated. Between each boring, augers will be decontaminated using a portable steam cleaner. The following steps will be used to decontaminate drilling and heavy equipment:

- Equipment having drilling cuttings caked on will be scraped off with a flat-bladed scraper.
- Equipment that will not be damaged by water, such as drill rigs, augers, drill bits, and tools will be sprayed by a high-pressure steamer.
- Decontamination will continue until all equipment is devoid, both inside and out of any materials, such as grease, gravel, and soil.

Drilling equipment will be decontaminated prior to drilling, between each boring, and prior to leaving the site. Decontamination will occur at the designated decontamination area. Following decontamination, drilling equipment will be placed on the clean drill rig and moved to a clean area. If the equipment is not used immediately, it should be stored in a designated secure, clean area.

5.3 DOCUMENTATION

Sampling personnel will be responsible for documenting the decontamination of sampling and drilling equipment. The documentation will be recorded with waterproof ink in the sampler's field notebook with consecutively numbered pages. The information entered in the field book concerning decontamination should include the following:

- Decontamination personnel
- Date and start and end times
- Decontamination steps/observations
- List of equipment decontaminated
- Weather conditions

APPENDIX J-2
SOP NO. 2 - GROUNDWATER MONITORING WELL AND PIEZOMETER INSTALLATION

1.0
TITLE PAGE

STANDARD OPERATING PROCEDURE NO. 2

GROUNDWATER MONITORING WELL AND PIEZOMETER INSTALLATION

LAMEFOOT MINE SITE

2.0
TABLE OF CONTENTS

Section | Page

1.0 TITLE PAGE 1

2.0 TABLE OF CONTENTS 2

3.0 PURPOSE AND SCOPE 3

4.0 PROCEDURES FOR DRILLING AND MONITORING WELL INSTALLATION 4

- 4.1 LIST OF NECESSARY EQUIPMENT 4
- 4.2 DRILLING AND WELL INSTALLATION PROCEDURES 5
  - 4.2.1 Drilling Techniques 5
  - 4.2.2 Stratigraphic Logging 6
  - 4.2.3 Well Materials Specifications 7
  - 4.2.4 Measurements 10
  - 4.2.5 Well Installation Procedure 10
- 4.3 WELL DEVELOPMENT 15
- 4.4 DOCUMENTATION 16

LIST OF FIGURES

FIGURE 1 WELL CASING SCHEMATIC FOR DEEP BEDROCK WELLS 19
FIGURE 2 WELL CASING SCHEMATIC FOR SHALLOW WELLS 20
FIGURE 3 EXAMPLE WELL LOG FORM 21

3.0

PURPOSE AND SCOPE

The purpose of this document is to define the standard operating procedure (SOP) for installing ground water monitoring wells and piezometers at the Lamefoot Mine Site. This SOP will explain the necessary equipment and procedures for well installation in both bedrock and alluvial situations. Certain geologic investigatory methods, such as packer testing, may be accomplished in the borehole if specified in the Hydrologic Monitoring Plan (HMP). The well locations will be specified in the HMP. The step-by-step procedures described herein are sufficiently detailed to allow field personnel to properly install wells.

This procedure is intended to be used together with the HMP and other SOPs. SOP No. 1 describes decontamination procedures which are applicable to this SOP.

4.0

PROCEDURES FOR DRILLING AND MONITORING WELL INSTALLATION

4.1 LIST OF NECESSARY EQUIPMENT

The following is an equipment list for well installation:

- PVC well casing and well screen (Schedule 40 and 80)
- Bentonite pellets
- Filter sand (#10-20 and #6-12)
- Cement and powdered bentonite for grouting (Portland cement Type II)
- Stainless steel centralizers
- Protective well casing with locking cap
- High-pressure steamer/cleanser
- Long-handled bristle brushes
- Wash/rinse tubs
- Alconox detergent
- Location map
- Drill rig capable of installing wells to the desired depth in the expected formation materials and conditions
- Plastic bags (ziploc)
- Self-adhesive labels
- Geophysical logging tools
- Weighted tape measure
- Water level probe
- Deionized water
- Appropriate health and safety equipment
- Log book

- Boring log sheets
- Well construction forms

4.2 DRILLING AND WELL INSTALLATION PROCEDURES

4.2.1 Drilling Techniques

Bedrock wells at the site may range from 150 to 650 feet deep. Wells in the unconsolidated sediments are expected to be less than 50 feet in depth. The preferred drilling method for installing wells in this setting is air rotary. With the air rotary method a tricone bit and percussion hammer are attached to the leading joint of drill pipe and rotated to advance the hole. Compressed air is routed downhole through the inside of the drill pipe exiting through the bit. Rock cuttings and formation water are forced up the hole through the annular space between the drill pipe and borehole wall. The key advantage of using this method is that semi-quantitative estimates of water production can be made as various geologic formations are penetrated. The reverse circulation method may also be used. A variety of geophysical logs and hydrogeologic tests can also be run in the uncased hole prior to well installation to aid in stratigraphic and hydrogeologic interpretation and correlation.

When drilling the deeper bedrock boreholes, permanent steel casing may need to be installed in the upper portion of the borehole (i.e., throughout the unconsolidated sediments) to prevent loss of circulation, aquifer cross-communication, and borehole collapse. Therefore casing will be set as needed to advance the hole to the planned depth. This may require that the casing and borehole size be telescoped, whereby a progressively small diameter casing is set with depth (inside the other) as the hole is advanced (See Figure 1). If it becomes apparent that the air rotary method with the permanent steel casing is not effective in removing the cuttings, keeping the borehole open, or preventing circulation loss, the cuttings will be removed from the borehole using foam or a biodegradable drilling additive (e.g., guar gum). Drilling mud will be avoided unless this is the only alternative left.

Wells in the unconsolidated sediments, which are anticipated to be less than 50 feet deep, will be installed using air rotary or dual tube percussion methods. Well completion details are shown in Figure 2.

4.2.2 Stratigraphic Logging

Borehole stratigraphy will be logged by examination of the sample cuttings. The data will be recorded in the field book and will include the following information:

- Project name and number
- Drilling company name
- Date drilling started and finished
- Type of bit and size
- Casing sizes and depths
- Well completion details
- Driller's name
- Geologist's name
- Type of drill rig
- Boring number
- Surface elevation (if available)
- Sample depths and times
- Sample characteristics with depth, such as lithology, grain size, sorting, texture, structure, bedding, color, moisture content, and the Unified Soil Classification (if in unconsolidated geologic materials)
- Water levels
- Drilling observations
- Other pertinent information

Cuttings will be collected at a minimum of 5-foot intervals, or at changes in lithology, and described by the field geologist or engineer. No soil or rock samples will be collected for chemical analysis.

4.2.3 Well Materials Specifications

Well Casings

Well casing will consist of new, threaded, flush-joint, 2-in ID, schedule 40 PVC for wells less than 100 feet in total depth, and Schedule 80 PVC for wells greater than 100 feet deep. Joints within the casing string will be threaded. The tops of all well casings will be fitted with caps, which can be easily removed by hand.

Casing centralizers will be stainless steel and will be firmly attached to the casing. Centralizers will be attached at the base and top of the well screen and at minimum 100-foot intervals on the casing for the deep bedrock wells. All well casing will be free of foreign material and steam-cleaned with approved water before use.

Well Screens

Well screens will consist of new 2-in. ID threaded PVC with factory-machined slots for wells in unconsolidated and bedrock materials. Screens used in unconsolidated sediment and bedrock wells will have a slot size of 0.02 inches. In no case will screen length exceed 20 feet in a given well. The screen may be shorter than 20 feet in the shallow wells depending on field conditions. All well screens will have an inside diameter equal to or greater than that of the well casing. The schedule thickness of PVC screen will be the same as that of the well casing. All screen bottoms will be fitted with a 2-foot sump or cap of the same composition as the screen.

Filter Pack

The filter pack material for the shallow unconsolidated sediment and bedrock wells will respectively consist of a #10-20 and #6-12 silica sand or equivalent. The filter pack will extend at least 3 feet but no more than 5 feet above the top of the well screen.

In borings where the screened interval is within a highly fractured zone, installation of a filter pack may not be possible. In this case the well completion design will be modified as necessary to best facilitate the purpose of the well. This may include use of a prepacked casing or no filter pack at all.

Bentonite Seal

A bentonite seal shall be installed above the filter pack. The seal will consist of a layer of commercially available bentonite pellets that is approximately 3 to 7 feet thick as measured immediately after placement, without allowance for swelling. Bentonite slurry seals should be used only when bentonite pellets cannot be placed in the annulus and the slurry should have a thick batter-like consistency. Slurry seals will be emplaced by tremie method and will have a maximum placement thickness of 7 feet.

In borings where no filter pack is possible the bentonite seal will be placed above a packer or similar device set in competent rock at least 5 feet above the screened interval.

Cement/Bentonite Grout

The annular space between the well casing and the borehole will be grouted from the top of the bentonite seal to the ground surface. In the case of the deeper bedrock wells, if the grout seal will be placed on top of a bentonite slurry seal, a period of 2 hours should be allowed between the placement of the bentonite and the grout seals. If a bentonite pellet seal has been used, a period of 1 hour should be allowed before placing the grout seal. For the shallow glaciofluvial wells, a period of 1/2-hour should be allowed between the placement of the bentonite and grout seals. The grout mixture will consist of approximately ten parts cement (Portland cement, type II) to one-half part bentonite (equivalent to one 94-pound bag of cement to about 5 pounds of bentonite powder) with approximately 8 gallons of water per 94-pound bag of cement. The grout will be prepared in an above-ground rigid container by first thoroughly mixing the cement with water and then mixing in the bentonite powder. For deep holes where large volumes of grout are necessary, the grout may be mixed off-site and transported to the site by truck. In addition for deeper holes, grout will be placed in two lifts. The first lift will be placed in the first 100 feet above the bentonite seal, and will be allowed to set for approximately 24 hours before pouring the second lift. The remainder of the well annulus will be grouted in the second lift.

Grout will be placed in the well annulus with a tremie pipe located approximately 10 feet above the top of the bentonite seal. The grout will be pumped through the pipe which will be pulled up incrementally until undiluted grout flows from the annular space at ground surface.

Above Ground Completion

Following initial annular grout placement, a minimum 8-inch (I.D.), 5-foot long, protective steel casing with hinged and lockable steel cap shall be installed over the monitoring well casing which projects above the ground surface.

Concrete Collar

An internal/external mortar collar will be placed within the protective casing annulus from ground surface to 0.5 foot above ground surface. The mortar mix will be one part cement to two parts sand, by weight, with minimal water for placement. The external mortar collar will extend approximately 3 feet radially from the protective casing to provide an effective surface seal.

4.2.4 Measurements

Measurements made in the boring during drilling and well construction, and of the well construction materials will be made to the nearest 0.1 foot. Following well completion the well location will be surveyed by a licensed surveyor. The horizontal location will be measured to 0.1 foot and will be tied to the local, documentable coordinates. The elevation of the top of the protective casing, ground surface, and water level measuring point will be measured to 0.01 foot and will be tied to a documentable bench mark.

4.2.5 Well Installation Procedure

Bedrock Wells

The procedure for bedrock well installation is as follows:

1) Decontaminate all drilling equipment according to procedures outlined in SOP No. 1.

2) Advance boring through surficial unconsolidated material to the bedrock contact using a nominal 8-in. tricone bit and collect cuttings at 5-foot intervals.

3) Set nominal 8-inch surface casing and grout the annular space using a pressure grouting method if necessary.

4) Allow grout to harden.

5) Advance boring to planned depth using a nominal 6-inch bit. Collect drill cuttings at 5-foot intervals.

6) Measure depth of completed boring using a weighted tape or measured drill pipe sub and bit.

7) Plug back boring to desired depth only if necessary (i.e., no deeper water bearing units) using grout filter sand capped with a minimum 5-foot thickness of bentonite chips or pellets.

8) Decontaminate all well materials according to SOP No. 1.

9) Measure each joint of casing, screen, and sump to nearest 0.10 foot. Attach packer device at appropriate interval if no filter pack will be used.

10) Assemble screen and casing as it is lowered into the well bore attaching stainless steel centralizers as required (Section 4.2.5).

11) Lower screen and casing until sump rests on bottom of borehole.

12) Record level of top of casing and calculate screened interval. Adjust screen interval by raising assembly to desired interval if necessary.

13) Calculate volume of filter pack, bentonite seal and grout required for existing borehole conditions and record.

14) Begin adding filter pack sand around the annulus of the casing in 5-foot increments. Repeated depth soundings shall be taken to monitor the level of the top of the sand in the hole. If no filter pack sand is used in the well, establish the bentonite seal above the screened interval.

15) Allow sufficient time for the filter sand to settle through the water column outside the casing before measuring the sand level.

16) Extend the filter pack sand to approximately 3 to 5 foot above the top of the well screen.

17) Following sand pack placement, install a minimum 5- to 7-foot-thick seal of bentonite pellets by slowly adding the pellets to avoid bridging. In general, install the seal using this method if the water column through which the pellets are dropping is less than 100 feet. The thickness of the completed bentonite seal shall be measured before the pellets are allowed to swell. The completed bentonite seal shall be allowed to hydrate for approximately 60 minutes before proceeding with the grouting operation.

If the water column through which the bentonite seal is to be placed exceeds 100 feet, consideration should be given to placing a bentonite slurry seal (minimum of 5- to 7-feet thick) consisting of a pure bentonite powder and potable water mix using a pump and tremie pipe. Grouting operations can continue 2 hours after placement of the bentonite slurry seal.

18) Grout the remaining annulus from the top of the bentonite seal to the ground surface after placing the bentonite seal. Grouting will be accomplished using two lifts. In the first lift, the first 100 feet above the bentonite seal will be grouted, and will be allowed to set for approximately 24 hours before grouting the remainder of the annulus in the second lift. The grout will be tremied

into the borehole until the annulus is completely filled. The base of the tremie pipe should be placed approximately 10 feet above the bentonite seal and incrementally pulled up as grouting proceeds.

19) Before the grout sets, center the protective steel casing on the well casing and insert into the grouted annulus. Insert a temporary spacer between the protective casing locking lid and the well cap to prevent the protective casing from settling onto the well cap. Place the well number, depth, and date on the well casing and cap.

20) After grout sets for 24 hours it will be checked for settlement. Additional grout will be added to top off the annulus, if necessary.

21) After 24 hours, install the mortar collar according to specifications in Section 4.2.5.

Shallow Wells

The procedure for shallow well installation using air rotary or the dual tube method is as follows:

1) Decontaminate all drilling equipment according to procedures outlined in SOP No. 1.

2) Advance boring to planned depth using nominal 6- to 8-inch bit. Collect drill cuttings at 5-foot intervals.

3) Measure depth of completed boring using a weighted tape.

4) Decontaminate all well materials according to SOP No. 1.

5) Measure each joint of casing, screen, and sump to nearest 0.10 foot.

6) Assemble screen and casing as it is lowered into the well bore.

7) Lower screen and casing until sump rests on bottom of borehole.

8) Record level of top of casing and calculate screened interval. Adjust screen interval by raising assembly to desired interval if necessary and add bentonite pellets or sand to come to base of sump.

9) Calculate volume of filter pack, bentonite seal and grout required for existing borehole conditions and record.

10) Begin adding filter pack sand around the annulus of the casing in 5-foot increments. Repeated depth soundings shall be taken to monitor the level of the sand.

11) Allow sufficient time for the filter sand to settle through the water column outside the casing before measuring the sand level.

12) Extend the filter pack sand to approximately 3 feet above the top of the well screen.

13) Following sand pack placement, install a minimum 3-foot-thick seal of bentonite pellets by slowly adding the pellets to avoid bridging. The thickness of the completed bentonite seal shall be measured before the pellets are allowed to swell. The completed bentonite seal shall be allowed to hydrate for approximately 30 minutes before proceeding with the grouting operation.

14) Grout the remaining annulus from the top of the bentonite seal to the ground surface after placing the bentonite seal. The grout will be tremied into the borehole until the annulus is completely filled. The base of the tremie pipe should be placed approximately 5 feet above the bentonite seal.

15) Before the grout sets, center the protective steel casing on the well casing and insert into the grouted annulus. Insert a temporary spacer between the protective casing locking lid and the well cap to prevent protective casing from settling onto the well cap. Place the well number, depth, and date on the well casing and cap.

16) After grout sets for 24 hours it will be checked for settlement. Additional grout will be added to top off the annulus, if necessary.

17) After 24 hours, install the mortar collar according to specifications in Section 4.2.5.

4.3 WELL DEVELOPMENT

After the well installation is complete, the wells will be developed to help repair the damage to the surrounding aquifer sediments induced by drilling and to promote groundwater flow from the aquifer into the well. If water was added to the well during construction or packer testing attempts will be made to remove the same volume of water added during the drilling. Other objectives of the well development program are to allow representative groundwater samples and accurate groundwater level measurements to be obtained, and to remove fine-grained sediment in the filter pack and nearby formation so that groundwater samples are not turbid and silting of the borehole wall does not occur.

Following well construction, each well will be developed to improve yield and minimize turbidity of the water. Wells will be developed using a submersible pump or bailer and a surge block or air lift methods. Well development will begin no sooner than 24 hours after the well has been grouted and will consist of removing approximately 3 to 10 well casing volumes from the well, plus a volume of water equal to any additional potable water added to the borehole during drilling or well installation. During well development, the field geologist or hydrogeologist will sample the development water for pH, temperature, specific conductance, and other observations (i.e., color, clarity, and turbidity) after each well casing volume has been evacuated. The well will be developed until the water clarity is relatively clear and/or all field parameters have stabilized (i.e., less than a 10 percent change between measurements). If for any reason this cannot be accomplished, the well will be purged of 10 well casing volumes. The field geologist or hydrogeologist will determine when the well has been adequately developed.

4.4 DOCUMENTATION

Observations and data acquired in the field during drilling and installation of wells will be recorded to provide a permanent record. These observations will be recorded with waterproof black ink in a bound weatherproof field book with consecutively numbered pages. Notes will be recorded daily when in the field. The information in the field book will include the following as a minimum:

- Project name and number
- Observer's name
- Drilling and well installation observations as described in Section 3.2
- Decontamination observations described in SOP No. 1
- Weather conditions

Field books will be kept by field personnel during the project. The field books will be used daily by the field personnel.

A well log (Figure 3) will be completed for each boring with the observations recorded in the field book. The well installation specifics will be shown in the well log and noted in the field book. Each well diagram will consist of the following (denoted by depth from ground surface):

- Bottom of the boring
- Casing depth (if intermediate casing is left in the hole)
- Screen location(s)
- Coupling/joint locations
- Filter pack
- Bentonite seals
- Cave-in locations
- Centralizers
- Height of riser without cap (above ground surface)
- Protective casing detail

Additional documentation for well construction in the field book will include the following:

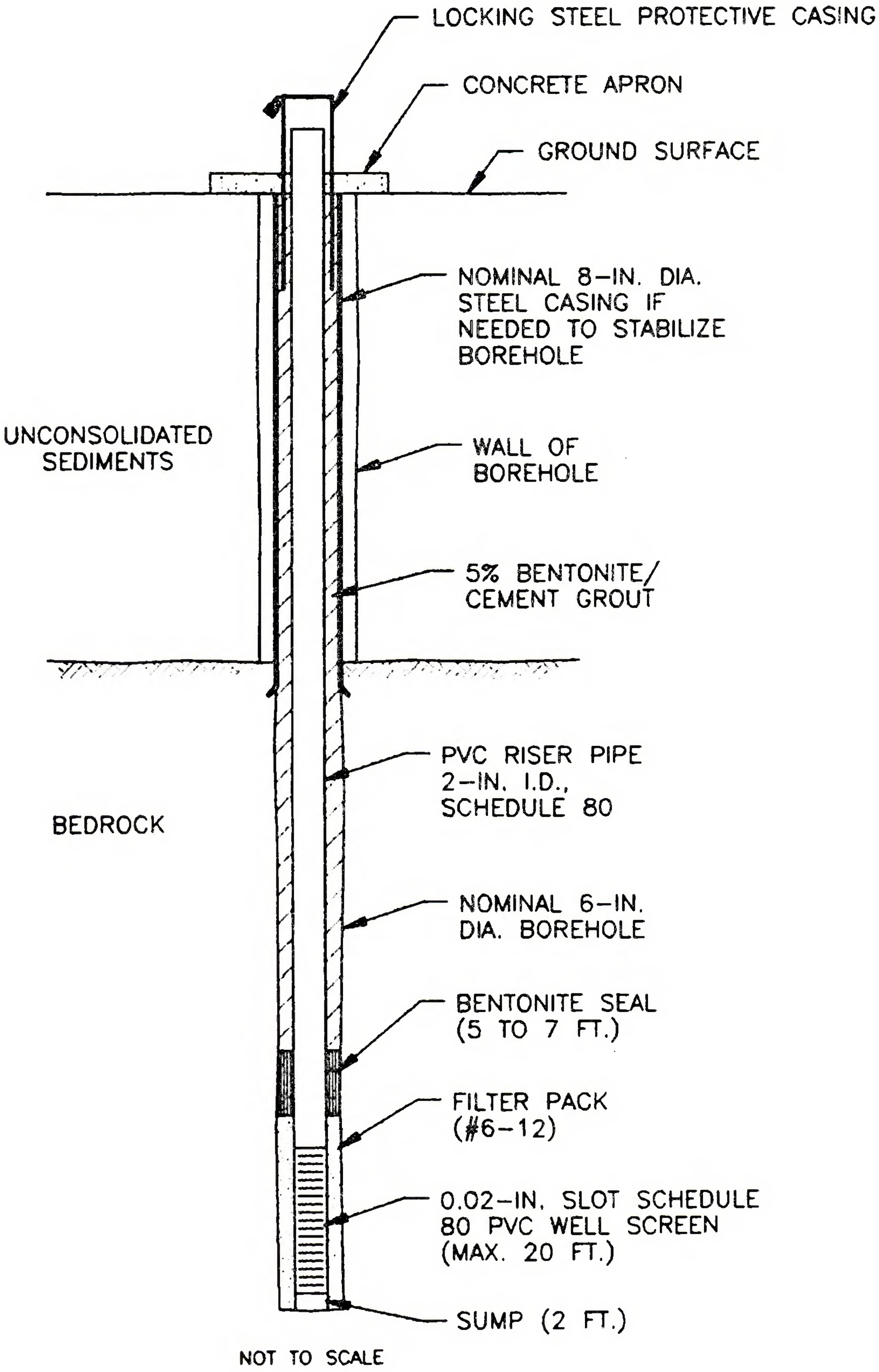
- Grout, sand, and bentonite volume calculations prior to well installation.

- The quantity and composition of the grout, seals, and filter pack actually used during construction.

- Screen slot size (in inches), slot configuration, total open area per foot of screen, outside diameter, nominal inside diameter, schedule/thickness, composition, and manufacturer.

- Coupling/joint design and composition.

- Centralizer design and composition.
- Protective casing composition and nominal inside diameter.
- Start and completion dates.
- Discussion of all procedures and any problems encountered during drilling and well construction.
- Surface completion information and date.


LOCKING STEEL PROTECTIVE CASING
CONCRETE APRON
GROUND SURFACE
NOMINAL 8-IN. DIA. STEEL CASING IF NEEDED TO STABILIZE BOREHOLE
UNCONSOLIDATED SEDIMENTS
WALL OF BOREHOLE
5% BENTONITE/ CEMENT GROUT
PVC RISER PIPE 2-IN. I.D., SCHEDULE 80
BEDROCK
NOMINAL 6-IN. DIA. BOREHOLE
BENTONITE SEAL (5 TO 7 FT.)
FILTER PACK (#6-12)
0.02-IN. SLOT SCHEDULE 80 PVC WELL SCREEN (MAX. 20 FT.)
SUMP (2 FT.)
NOT TO SCALE

Job No. : 23295-FEIS

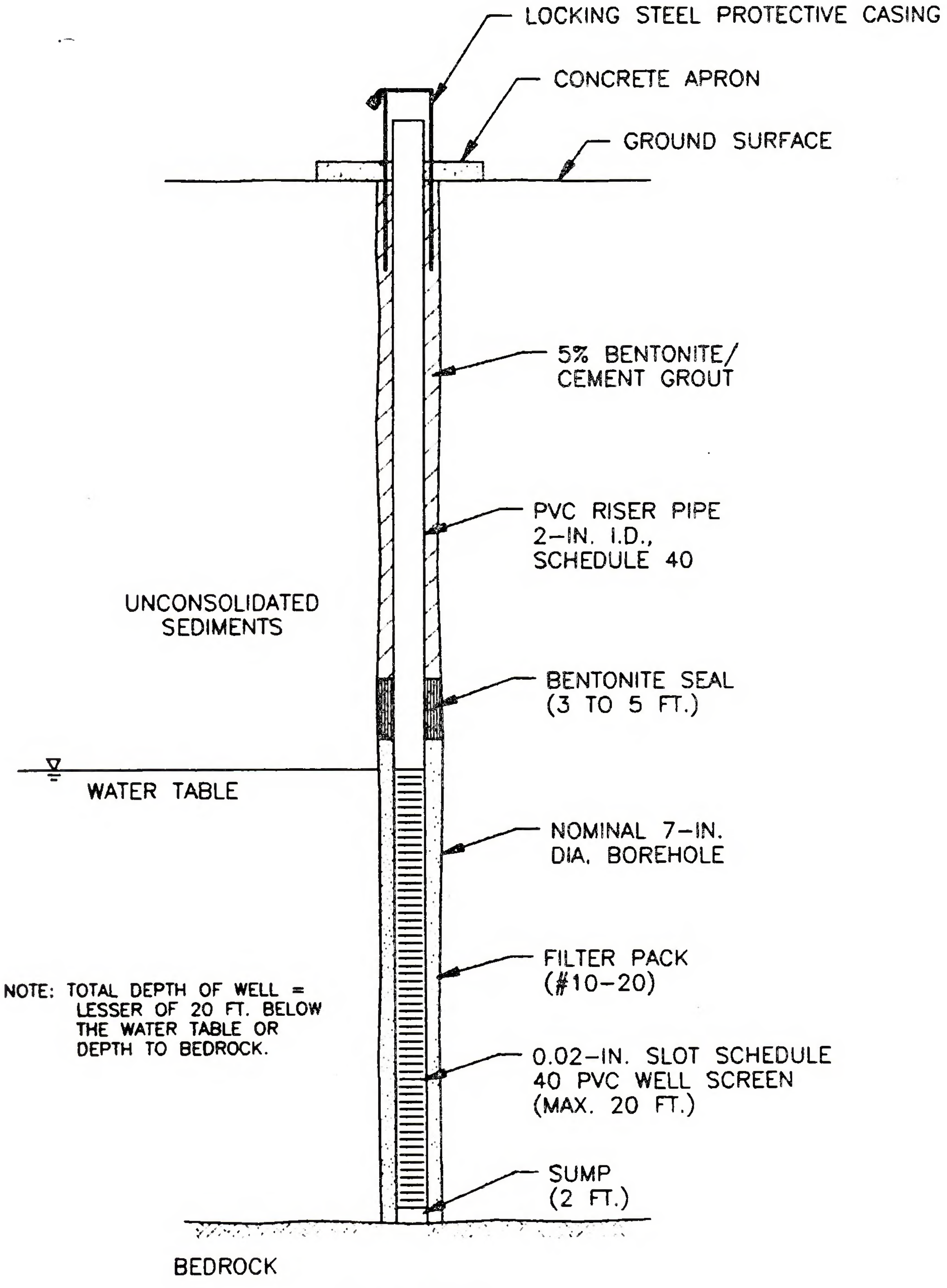
Prepared by : G.B.K.

Date : 4/22/94

WELL CASING SCHEMATIC
FOR DEEP BEDROCK WELLS

23295WC2

FIG. 1


LOCKING STEEL PROTECTIVE CASING
CONCRETE APRON
GROUND SURFACE
5% BENTONITE/ CEMENT GROUT
PVC RISER PIPE 2-IN. I.D., SCHEDULE 40
UNCONSOLIDATED SEDIMENTS
BENTONITE SEAL (3 TO 5 FT.)
WATER TABLE
NOMINAL 7-IN. DIA. BOREHOLE
FILTER PACK (#10-20)
0.02-IN. SLOT SCHEDULE 40 PVC WELL SCREEN (MAX. 20 FT.)
SUMP (2 FT.)
BEDROCK

NOTE: TOTAL DEPTH OF WELL = LESSER OF 20 FT. BELOW THE WATER TABLE OR DEPTH TO BEDROCK.

NOT TO SCALE

Job No. :	23295-FEIS
Prepared by :	G.B.K.
Date :	4/22/94

WELL CASING SCHEMATIC
FOR SHALLOW WELLS

23295WC1

FIG. 2

FIGURE 3: EXAMPLE WELL LOG FORM

Project:

Well No.

North:
East:

Date Started:		Total Depth:	Casing Elev:	GW ATD:	
Date Completed:		Perforation:	☰ From	to	
Logged By:	Checked By:	Pack:	░ From	to	
Drilling Co:	Driller:	Seal:	▢ From	to	
Drilling Method:			▨ From	to	
		Casing:	Drill Bit Diameter:		
Drilling Equipment:		Sampler:			

Depth (feet)	LITHOLOGIC DESCRIPTION	LITH.	WELL CONSTRUCTION	Sample	Blows Per 6"	REMARKS
5						
10						
15						
20						
25						
30						

Project: **Woodward-Clyde Consultants** Page 1 of

Project:	Well No.	North: East:

Depth (feet)	LITHOLOGIC DESCRIPTION	LITH.	WELL CONSTRUCTION	Sample	Blows Per 6"	REMARKS
40						
45						
50						
55						
60						
65						
70						
75						

APPENDIX J-3
SOP NO. 3 - FIELD PACKER TESTING

1.0
TITLE PAGE

STANDARD OPERATING PROCEDURE NO. 3

FIELD PACKER TESTING

LAMEFOOT MINE SITE

2.0
TABLE OF CONTENTS

Section		Page
1.0	TITLE PAGE	1
2.0	TABLE OF CONTENTS	2
3.0	INTRODUCTION	3
4.0	RESPONSIBILITIES AND QUALIFICATIONS	4
5.0	PROCEDURES FOR CONDUCTING PACKER TESTS	5
	5.1 PROCEDURES	5
	5.1.1 Hole Preparation	5
	5.1.2 Packer Test Procedures	5
6.0	REFERENCES	14

LIST OF FIGURES

FIGURE 1	PACKER TEST SET-UP
FIGURE 2	DRILL ROD ARRANGEMENT TO CALIBRATE HEAD LOSS IN DRILL RODS
FIGURE 3	DRILL ROD CALIBRATION DATA SHEET
FIGURE 4	CALIBRATING PRESSURE LOSSES IN DRILL RODS/WATER PIPE
FIGURE 5	PACKER TEST DATA SHEET (METHOD A)
FIGURE 6	TYPICAL PRESSURE VERSUS FLOW CURVES FOR PACKER TESTS
FIGURE 7	PUMP-IN TEST ARRANGEMENT USING PNEUMATIC PACKERS
FIGURE 8	MONITORING PACKER LEAKAGE IN PACKER TESTS
FIGURE 9	PACKER TEST DATA SHEET (METHOD B)

3.0

INTRODUCTION

The purpose of this document is to define the Standard Operating Procedure (SOP) for conducting packer tests in boreholes at the Lamefoot Mine Site. This procedure is intended to be complete enough to allow field personnel to properly conduct a packer test.

This SOP describes the needed equipment and field procedures necessary to conduct packer tests.

This SOP is to be strictly followed, and any modification to the procedure shall be approved by the Project Manager or Task Leader in advance.

4.0

RESPONSIBILITIES AND QUALIFICATIONS

The Project Manager (Echo Bay's or consultant's) or Task Leader is responsible for assigning project staff to conduct packer tests. The Task Leader is also responsible for assuring that this and any other appropriate procedures are followed by project personnel.

The project staff assigned to the conductance of packer tests are responsible for completing their tasks according to this and other appropriate procedures. All staff are responsible for reporting deviations from the procedure or nonconformance to the Task Leader or Project Manager.

Only qualified personnel shall be allowed to perform this procedure. Qualifications may be based on previous experience or on-the-job training and supervision by another qualified person.

5.0
PROCEDURES FOR CONDUCTING PACKER TESTS

5.1 PROCEDURES

5.1.1 Hole Preparation

Packer testing will be conducted at multiple intervals in the bedrock at selected boreholes. A rotary drilling rig will be used to drill from the ground surface through the alluvium to the bedrock contact. A surface casing extending to the bedrock contact will then be set. Drilling will then be performed in the bedrock to the specified total boring depth. If necessary, the drill hole will be flushed of drill cuttings by blowing air or water through the drill string prior to conducting the packer tests. It is anticipated that packer tests will be conducted in discrete 20 feet intervals throughout the borehole, requiring the use of two packers to isolate the tested interval.

5.1.2 Packer Test Procedure

Two methods for conducting packer tests are presented. The primary difference between the two methods is that Method A requires calibration of friction head losses in the system to estimate the effective fluid pressure in the test cavity, while Method B uses a sensing device in the test section to monitor the cavity pressure directly. Method A is presented in Section 5.1.2.1 while Method B is presented in Section 5.1.2.2.

5.1.2.1 Packer Test Method A

The following equipment is needed to perform Packer Test Method A:

- Drill rig or equivalent to lower and raise the water pipe and packers.

- Water meter, range 1 to 8 inches, depending on the permeability of the test zone and surface area of the test section. Generally 1-inch and 4-inch meters will be appropriate.

- Pressure gauge, range (in psi) to at least 1.5 times test depth (in feet).

- Pump, centrifugal-type, capacity to 350 gpm is adequate for most tests.

- Two packers.

- Drill rods or 1-1/4-inch diameter water pipe.

- Potable water supply with suction pump and hose and storage tank.

- Swivel and hose to pump; swivel connects hose to drill rods or water pipe.

- Compressed air and air hose if air-inflatable packers used.

The United States Bureau of Reclamation (1951) recommends arranging packer test equipment as follows, beginning at the water source:

> "Source of water, suction line, pump, water line to settling and storage tank or basin, suction line, centrifugal test pump, line to water meter adapter if required, or to water meter, short length of pipe, plug valve, water line to swivel, sub for gauge, pipe, or rod to packer. All connections should be kept as short and straight as possible with a minimum number of changes in diameter of hose, pipe, etc."

A schematic diagram of the packer test set-up is shown in Figure 1.

The test procedure for Method A is as follows:

1. Prepare the borehole as discussed in Section 5.1.1.

2. Determine the head loss in the drill rods or water pipe as follows. Assemble the drill rods (water pipe), swivel, pressure gauge, water meter and pump at ground level. Fit the last drill rod with a pressure gauge, a short length of pipe, and a valve and hose as shown in Figure 2.

   Pump at a range of expected flow rates and compute the head loss between the two pressure gauges for each flow rate. Use the valve at the end of the drill rods to control the flow. The maximum flow rate used should deliver a maximum pressure of 0.5 psi/ft depth to the test section. For example, for the length of drill pipe (water pipe) corresponding to a test depth of 40 feet, the maximum flow rate should produce a pressure of 20 psi in the gauge at the far end of the system. Other test flow rates can be percentages of the maximum value determined. Test pressures above 0.5 psi/ft depth may fracture the formation, resulting in an overestimation of the true permeability.

   Repeat this process over the range of drill rod lengths required for anticipated test depths. Curves are available to estimate head loss per 10 foot length of drill rod but they do not include the effects of the swivel, pipe connections, and individual characteristics of the drill rods (water pipes) used. The longer the drill rods, the greater the head loss and the more important it becomes to calibrate the equipment (Little et al., 1963). Dick (1975) stresses the importance of careful, routine calibration of equipment which conducts the water from the point of pressure measurement to the section of borehole being tested. The accuracy of packer test results depends significantly on the correct estimation of the head loss in the system.

3. Record the data in Step 2 on a log similar to that shown in Figure 3. The data may be plotted in either of two ways, depending on personal preference. Plot head loss (vertical scale) vs. flow, connecting the data points according to the length between the pressure gauges, or plot head loss (vertical scale) vs. the length of the system, connecting the data points according to flow rate. Since it may prove difficult to obtain consistent flow values throughout the calibration procedure, the former plot is the easier of the two to develop. The plot will be used in the analysis of the packer tests.

4. Seat the packers so that a test section of approximately 20 feet is obtained. The seating of the packers can be determined in conjunction with geologic information, or according to a pre-specified interval. For example, if a 4-foot- thick fractured section is sandwiched in massive rock, it should be isolated with packers and pressure tested. A complete profile can be obtained by repositioning the packers at intervals equivalent to the test zone length.

5. Pump water into the test section at a specified constant pressure for 15 to 20 minutes, taking readings of total water flow and pressure at 5 minute intervals (USBR 1951). The test at this pressure is completed when the flow over two successive 5 minute intervals differs by less than 10 percent. In very permeable materials, a test duration of 5 or 10 minutes may be sufficient. In this case, record flow and pressure over 1 minute intervals until stable conditions are reached. Perform this procedure at varying applied water pressures. A recommended test pressure sequence is $1/2P_{max}$, $3/4\ P_{max}$, P_{max}, $3/4\ P_{max}$, and $1/2\ P_{max}$, where P_{max} is defined as follows:

$$P_{max} = P_t + P_l$$

where P_t is the maximum allowable excess pressure at the test section, typically 0.5 psi/ft depth to the test section, but subject to any limitations on permissible packer pressures, and P_l is the pressure loss due to friction in the testing apparatus, determined using the procedure presented in Step 2, or estimated using Figure 4.

6. Record the data on a form similar to that in Figure 5.

7. Plot the data, pressure (vertical scale) vs. flow, and compare with Figure 6 for evidence of problems such as leakage around packer, erosion of test zone, and clogging of fissures. A high quality test should provide a linear plot similar to that corresponding to laminar flow in Figure 6.

8. For the stage test procedure, remove the packers and prepare the borehole for the next test.

5.1.2.2 Packer Test Method B

As previously discussed, an essential step in the Method A test procedure is the careful calibration of test equipment prior to testing. The desired data in a packer test are the flow rate and excess pressure applied to the given test section. Calculation of the true excess pressure at depth, as performed on the data sheet in Figure 5, requires the pressure loss in the packer stem (drill rod or water pipe) due to friction. In practice, short cuts are often taken, calibration is not performed regularly, and one set of "standard" calibration curves is used for all tests. Such practice can lead to significant error in permeability tests. Dick (1975) discusses many sources of variable head loss, including flexible hoses, fittings, placement of pressure meters, and variable age of the equipment.

As the depth of the testing increases, the frictional head loss in the packer stem increases significantly. It is at large depths where equipment calibrations are most crucial. The amount of time and effort required to calibrate the system increases with the increased stem length, as does the temptation to "standardize" calibration curves. Also, areal constraints imposed by a particular test site may make calibration impractical.

It is with this calibration difficulty in mind that a second method is proposed, a method which replaces the need for calibrating friction losses entirely. This is achieved by using a sensing device, a pressure transducer, which is placed down the hole in the test section and measures the test cavity pressure directly. This device also provides a means to detect and eliminate possible packer leakage. Using Test Method A, this information is frequently only noticed after the fact, if at all, from interpretation of P-Q plots.

The following equipment is needed to perform Packer Test Method B:

- Drill rig or equivalent to lower and raise the water pipe and packers.
- Water meter, range 1 to 8 inches, depending on the permeability of the test zone and the surface area of the test section. Generally 1-inch and 4-inch meters will be appropriate.
- Pump, centrifugal type, capacity to 350 gpm is adequate for most tests.
- Two packers.
- 1-1/4" diameter water pipe.
- Potable water supply with suction pump, hose, and storage tank.
- Swivel and hose to pump; swivel connects hose to drill rods or water pipe.
- Compressed air and air hose for pneumatic packers.

- Pressure transducer, cables and read out unit.
- Optional continuous recorder for transducer.

Similar specifications as to equipment arrangement, set-up, pipe size, pump equipment, swivels and water meters apply to Test Method B as are described in Section 5.1.2.1 for Test Method A. Pneumatic packers are recommended for use with Test Method B. In-line pressure gauges are not required for Test Method B, but are useful as a secondary data source.

Various types of pressure transducers (electrical piezometers) are commercially available. Piezometer cable may be easily attached to the packer stem as the packer assembly is lowered into the borehole, or a spool can control the cable feed as the assembly is lowered. A typical test arrangement is given in Figure 7.

The test procedure for Method B is as follows:

1. Prepare the borehole as per Section 5.1.1.
2. Position the packers so that a test section of approximately 20 feet is obtained. The seating of the packers to define the test section length can be adjusted according to the geologic information obtained from core samples, or it can be adjusted according to a pre-specified interval of interest to be tested.
3. Prior to expanding the packers, record the pretest water pressure (P_o) (if below the water table) in the test cavity using the pressure transducer. The difference between this pressure and the cavity pressure measured during testing is the excess pressure (P_T) applied to the immediate ground mass. P_T

is the pressure that is plotted vs. the observed flow rate (Q) into the test section.

4. Evaluate possible packer leakage. This can be accomplished with a continuous reading instrument (chart recorder) that monitors the pressure in the test cavity. This procedure should be applied as follows:

   Commence inflow into the system and then incrementally increase the packer pressure. This will contribute to the removal of any large air pockets within the test cavity. As the packer pressure increases, leakage past the packers will decrease, the pressure in the test cavity will increase, and the flow rate into the test cavity (measured by the water meter at the top of the borehole) will consequently decrease.

   After a certain point, further increase of packer pressure will only create a temporary slight increase in cavity pressure at the instant the packer pressure is increased. This is a dynamic effect, and the cavity pressure will quickly return to a steady state value.

   Three graphs that illustrate typical relationship between packer pressure, test cavity pressure, and test cavity inflow throughout the packer sealing procedure are contained in Figure 8.

   Use caution to ensure that the excess test cavity pressure does not exceed 0.5 psi/ft depth to the test section during this procedure. Otherwise, hydraulic fractures may occur and significantly alter the subsequent test results as discussed earlier.

5. Pump water into the test section at a specified constant pressure for 15 to 20 minutes, and take readings of total water flow and pressure at 5-minute

intervals. The test at this pressure is completed when the flow over two successive 5-minute intervals differs by less than 10 percent.

In very permeable materials, a test duration of 5 or 10 minutes may be sufficient. In this case, record flow and pressure over 1-minute intervals until stable conditions are reached. Perform this procedure at various applied water pressures. The maximum excess pressure applied should be 0.5 psi/ft depth to the test section.

As an example of a recommended sequence of pressures, consider a test section depth of 100 feet, where the existing water table is 50 feet below the ground level. The pretest pressure in the test section is approximately 22 psi (62.4 lb/ft^3 x 50 ft divided by 144 in^2/ft^2). The maximum allowable excess pressure is 50 psi (0.5 psi/ft x 100 ft). Therefore, during the test, the maximum observable pressure is 72 psi (22 psi existing hydrostatic pressure plus the maximum allowable excess pressure of 50 psi). The value of 72 psi is called P_{max}, and a recommended pressure sequence is ½P_{max}, ¾P_{max}, P_{max}, ¾P_{max}, and ½P_{max}.

6. Record the data on a form similar to that in Figure 9, where the test section pressure (P_T) is directly measured by the transducer.

7. Plot the data as pressure (vertical scale) vs. flow. Compare the data with Figure 6 for evidence of problems such as: leakage around packers, erosion of test zone, and clogging fissures. A high-quality test should provide a linear plot similar to that corresponding to laminar flow in Figure 6.

8. Remove the packers and prepare the borehole for the next test.

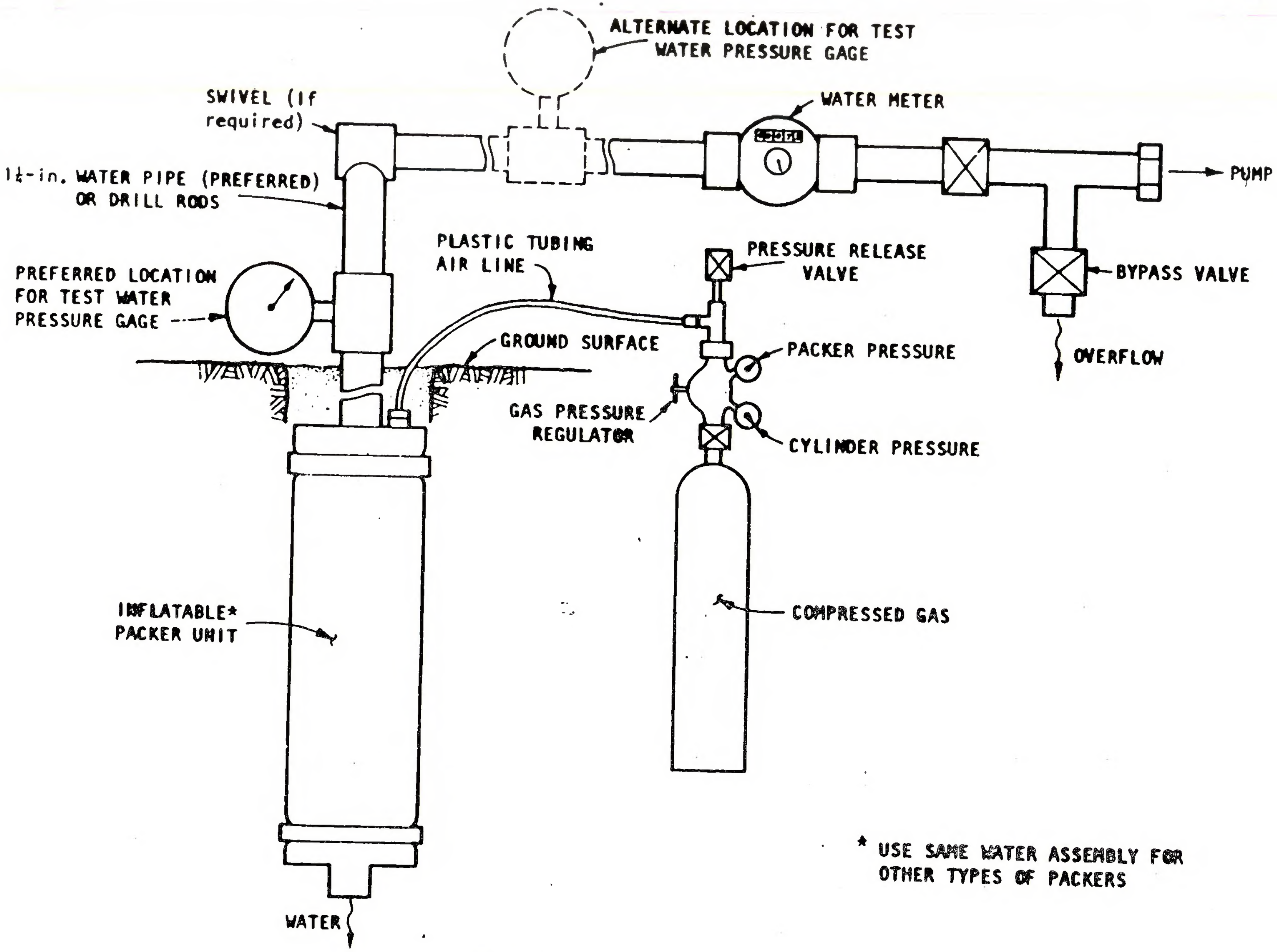
6.0
REFERENCES

Dick, R.C. 1975. "In-situ measurement of rock permeability: influence of calibration error on test results," AEG Bulletin, Vol. 12, No. 3.

Dixon, J.C. and K.B. Clarke. 1975. "Field Investigation Techniques" in Site Investigations in Areas of Mining Subsidence. F.G. Bell, editor. Newnes-Butterworths, London.

Little, A.L., J.C. Stewart and P.J. Fookes. 1963. "Bedrock grouting tests at Mangla Dam, West Pakistan," Grouts and Drilling Muds in Engineering Practice. Butterworths, London.

United States Bureau of Reclamation. 1951. "Permeability Tests Using Drill Holes and Wells," Geology Report G-97.



* USE SAME WATER ASSEMBLY FOR OTHER TYPES OF PACKERS

Figure 1 PACKER TEST SET-UP

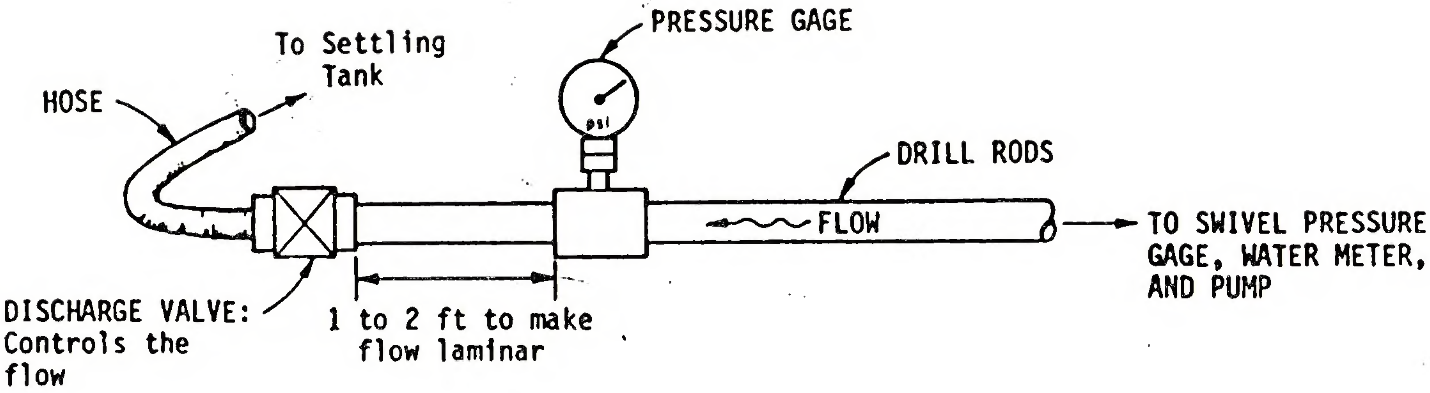

PRESSURE GAGE
To Settling Tank
HOSE
psi
DRILL RODS
FLOW
TO SWIVEL PRESSURE GAGE, WATER METER, AND PUMP
DISCHARGE VALVE: Controls the flow
1 to 2 ft to make flow laminar

Figure 2 DRILL ROD ARRANGEMENT TO CALIBRATE HEAD LOSS IN DRILL RODS

Figure 2

Drill Rod/Water Pipe Type ____

Nominal I.D. ____

Length of Hose between Upper Pressure Gage and Drill Rod ____

Swivel Type (if below pressure gage) ____

I.D. Swivel ____

Length of Drill Rod Between Lower Pressure Gage and Valve ____

Name ____

Date ____

Location ____

Borehole No. ____

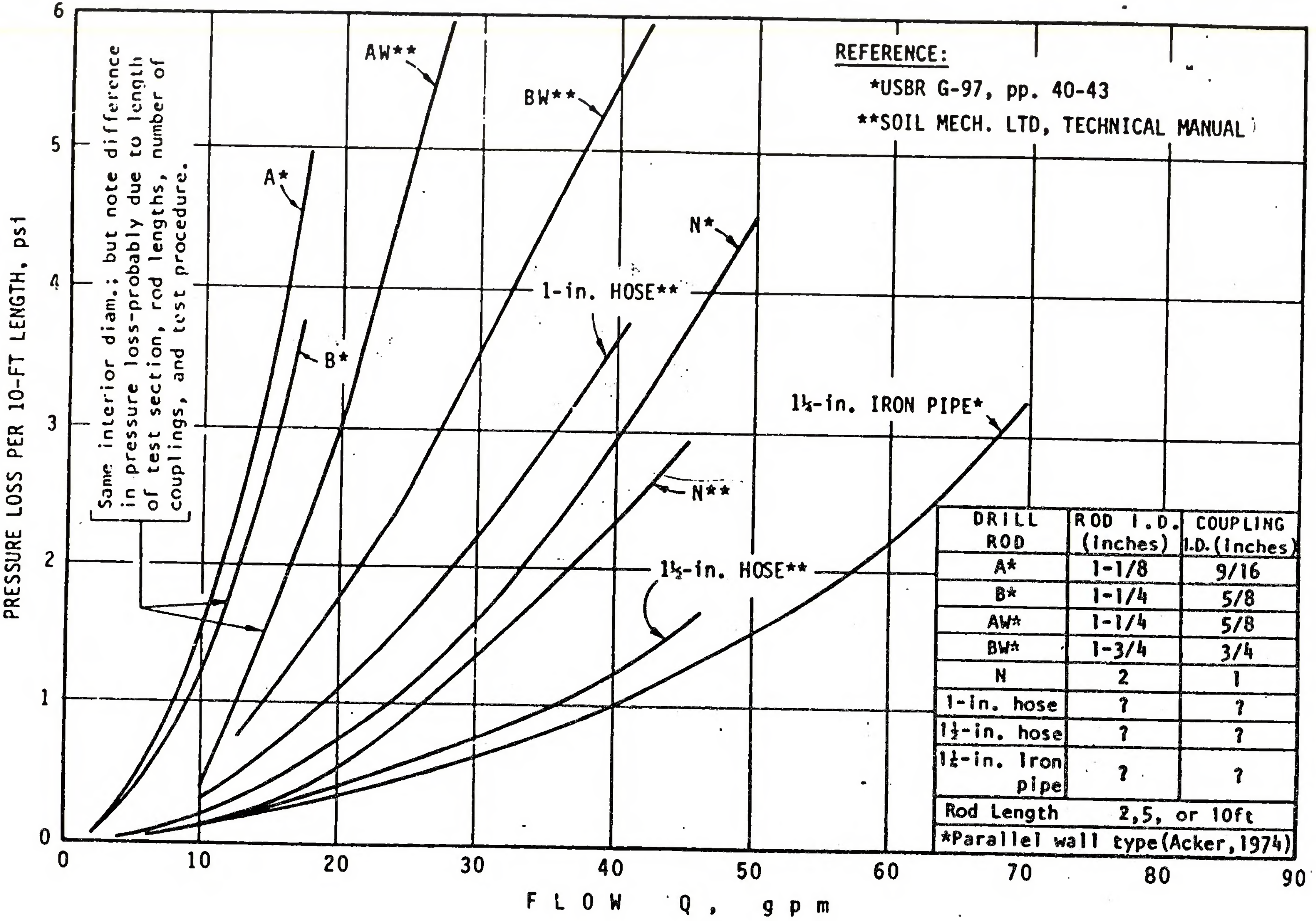
Nominal I.D. of Hose ____

Sketch drill rod/swivel/pressure gage arrangement

L	t	V_w	Δt	ΔV_w	Q	G_u	G_L	ΔG_t	$\Delta G_{\Delta t}$
Length of Drill Rod Between Gages	Time of Reading	Water Meter Reading at Time t	Time Between Readings	Change in Water Meter Reading Over Δt	Flow = $\Delta V_w/\Delta t$	Pressure at Upper Gage at Time t	Pressure at Lower Gage at Time t	Pressure Loss at Time t = $G_u - G_L$	Average Pressure Loss Over Interval Δt = $\Delta G_t - \Delta G_{t-1}$

Figure 3 DRILL ROD CALIBRATION DATA SHEET

Figure 3



REFERENCE:
*USBR G-97, pp. 40-43
**SOIL MECH. LTD, TECHNICAL MANUAL

Same interior diam.; but note difference in pressure loss-probably due to length of test section, rod lengths, number of couplings, and test procedure.

DRILL ROD	ROD I.D. (inches)	COUPLING I.D. (inches)
A*	1-1/8	9/16
B*	1-1/4	5/8
AW*	1-1/4	5/8
BW*	1-3/4	3/4
N	2	1
1-in. hose	?	?
1½-in. hose	?	?
1½-in. Iron pipe	?	?
Rod Length	2,5, or 10ft	
*Parallel wall type(Acker,1974)		

Figure 4 CALIBRATING PRESSURE LOSSES IN DRILL RODS/WATER PIPE

Location ______

Borehole No. ______

Depth of Bottom of Borehole ______

Depth of Top of Test Section ______

Depth of Base of Test Section ______

Depth of Casing ______

Diameter of Borehole ______

Type of Drill Rod/Water Pipe ______

I.D. of Drill Rod/Water Pipe ______

Length of Drill Rod ______

Packer Type ______

Packer Length (Upper) ______

Name ______

Date ______

Elevation ______

Depth to Ground Water ______

Elevation of Ground Water ______

Height of Pressure Gage above Ground Level ______

Length of Hose between pressure gage and drill rod ______

Nominal I.D. of hose ______

Swivel type ______

Nominal I.D. of swivel ______

If two Packers Used:

Packer Length (Lower) ______

Length of Test Section ______

Water pipe I.D. ______

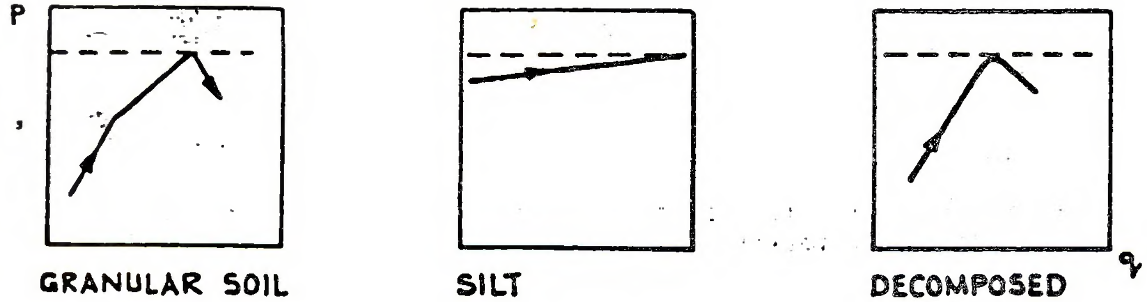
Area of pipe perforations ______

SYMBOL:	P_P	P_L	H_E	P_E	P_T	t	V_w	Δt	ΔV_w	Q	H_T
ITEM:	Water Gage Pressure	Pressure Loss in Drill Rods and Hose*	Static Head Gain: Elev. of Press. Gage - Elev. of Middle of Test Section	Pressure Gain due to H_E = $H_E \cdot \gamma_w$ Where γ_w = unit Weight of Water	Test Section Pressure = $P_E + P_P - P_L$	Time (min) From Start of Test	Water Meter Reading at Time t	Time (min) Between Readings	Change in Water Meter Reading Between Readings	Flow = $\Delta V_w / \Delta t$	Head at Test Section = P_T/γ_w γ_w = unit Weight of Water
UNITS:						min		min			

*If calibration test run, see Figure 3 Otherwise, see Figure 4

Figure 5 PACKER TEST DATA SHEET

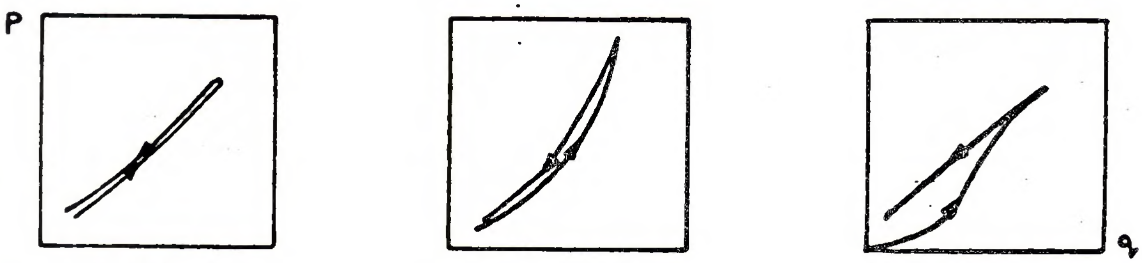
Figure 5



GRANULAR SOIL

SILT

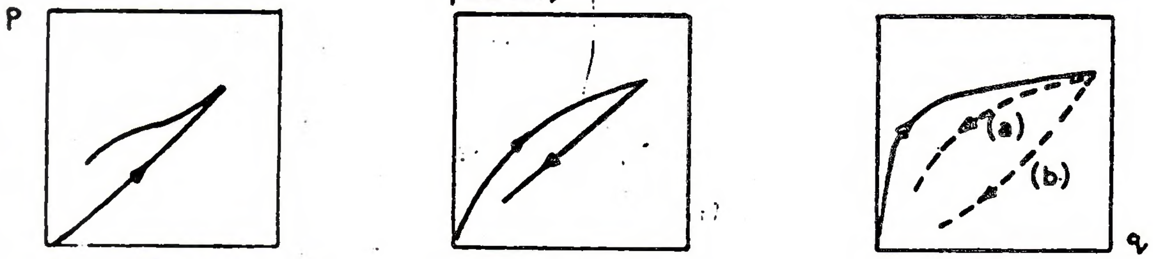
DECOMPOSED ROCK



LAMINAR FLOW

TURBULENT FLOW
(caused by a few open fissures or leakage past packer)

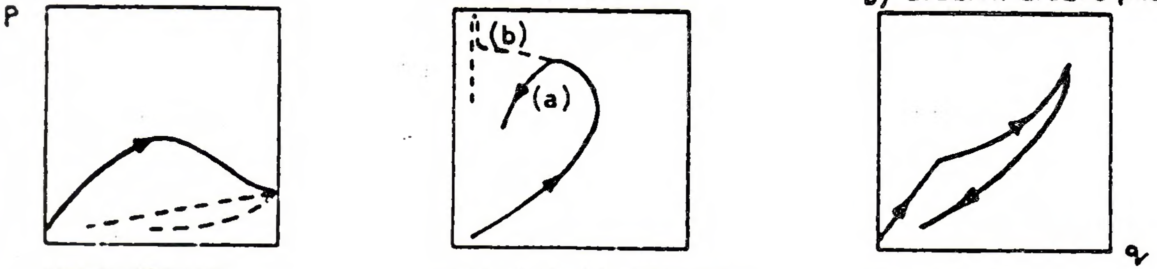
UPLIFT OF GROUND



RECOVERY OF GROUND

SCOUR OF FISSURE OR MOVEMENT OF PACKER AT HIGH PRESSURE.

MOVEMENT OF PACKER AT INTERMEDIATE PRESSURE
a) No erosion
b) Erosion around packer



PROGRESSIVE SCOUR OF FISSURE

SEALING OF FISSURES
a) partial
b) nearly complete

COMBINATION OF CIRCUMSTANCES.

Figure 6 TYPICAL PRESSURE VERSUS FLOW CURVES FOR PACKER TESTS (Dixon, 1975)

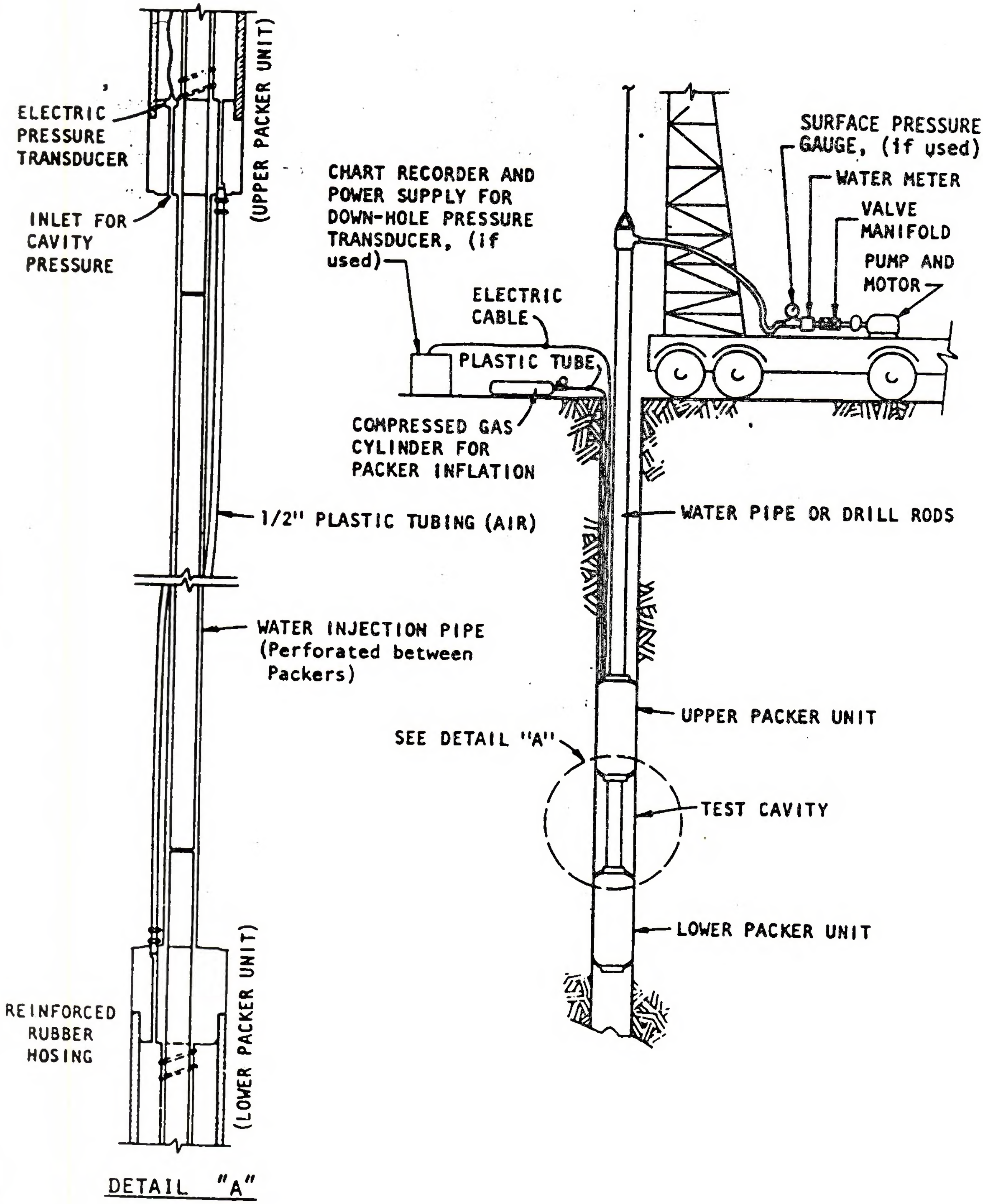
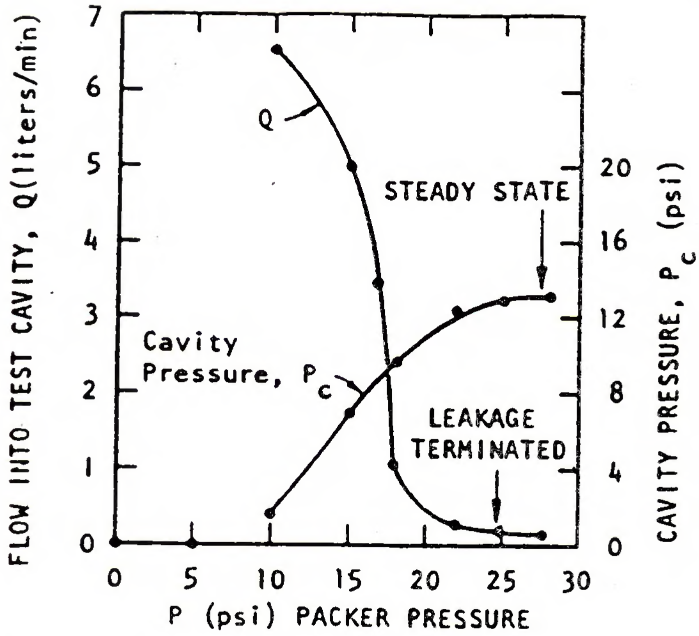
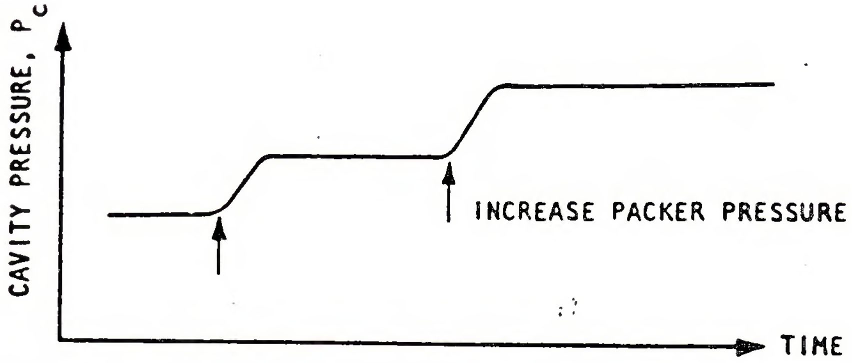

ELECTRIC PRESSURE TRANSDUCER
INLET FOR CAVITY PRESSURE
(UPPER PACKER UNIT)
1/2" PLASTIC TUBING (AIR)
WATER INJECTION PIPE (Perforated between Packers)
REINFORCED RUBBER HOSING
(LOWER PACKER UNIT)
DETAIL "A"
CHART RECORDER AND POWER SUPPLY FOR DOWN-HOLE PRESSURE TRANSDUCER, (if used)
ELECTRIC CABLE
PLASTIC TUBE
COMPRESSED GAS CYLINDER FOR PACKER INFLATION
SURFACE PRESSURE GAUGE, (if used)
WATER METER
VALVE MANIFOLD
PUMP AND MOTOR
WATER PIPE OR DRILL RODS
UPPER PACKER UNIT
SEE DETAIL "A"
TEST CAVITY
LOWER PACKER UNIT

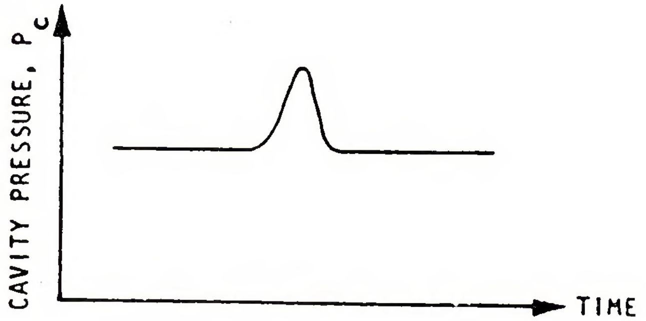
Figure 7 PUMP-IN TEST ARRANGEMENT USING PNEUMATIC PACKERS



(a) TYPICAL DATA INDICATING SEALING OF PACKER



(b) CHART RECORD OF TEST CAVITY PRESSURE WHILE INCREASING PRESSURE IN A LEAKY PACKER (Increases in Packer Pressure divert additional leakage into test cavity, thereby raising cavity pressure, P_c)



(c) CHART RECORD OF TEST CAVITY PRESSURE WHILE INCREASING PRESSURE IN A NON-LEAKING PACKER (Transient Peak in Cavity Pressure, P_c, due to increase in packer pressure indicates effective packer seal)

Figure 8 MONITORING PACKER LEAKAGE IN PACKER TESTS

(Maini, 1971)

Location ____________________
Borehole No. ____________________
Depth of Bottom of Borehole ____________________
Depth of Top of Test Section ____________________
Depth of Base of Test Section ____________________
Depth of Casing ____________________
Diameter of Borehole ____________________
Type of Drill Rod/Water Pipe ____________________
I.D. of Drill Rod/Water Pipe ____________________
Length of Drill Rod ____________________
Packer Type ____________________
Packer Length (Upper) ____________________

If two Packers Used:
Packer Length (Lower) ____________________
Length of Test Section ____________________
Water pipe I.D. ____________________
Area of pipe perforations ____________________

Name ____________________
Date ____________________
Elevation ____________________
Depth to Ground Water ____________________
Elevation of Ground Water ____________________
Height of Pressure Gage above Ground Level ____________________
Length of Hose between pressure gage and drill rod ____________________
Nominal I.D. of hose ____________________
Swivel type ____________________
Nominal I.D. of swivel ____________________

SYMBOL:	P_P	P_L	H_E	P_E	P_T	t	V_w	Δt	ΔV_w	Q	H_T
ITEM:	Water Gage Pressure	Pressure Loss in Drill Rods and Hose*	Static Head Gain: Elev. of Press. Gage - Elev. of Middle of Test Section	Pressure Gain due to H_E = $H_E \cdot \gamma_w$ Where γ_w = unit Weight of Water	Test Section Pressure = $P_E + P_P - P_L$	Time (min) From Start of Test	Water Meter Reading at Time t	Time (min) Between Readings	Change in Water Meter Reading Between Readings	Flow = $\Delta V_w/\Delta t$	Head at Test Section = P_T/γ_w γ_w = unit Weight of Water
UNITS:						min		min			

(P_L is determined from Figure 5)

Figure 9 PACKER TEST DATA SHEET

Figure 9

APPENDIX J-4

SOP NO. 4 - AQUIFER PUMPING TESTS

1.0

TITLE PAGE

STANDARD OPERATING PROCEDURE NO. 4

AQUIFER PUMPING TESTS

LAMEFOOT MINE SITE

2.0
TABLE OF CONTENTS

Section | Page

1.0 TITLE PAGE 1

2.0 TABLE OF CONTENTS 2

3.0 INTRODUCTION 3

4.0 RESPONSIBILITIES AND QUALIFICATIONS 4

5.0 PROCEDURE 5

- 5.1 EQUIPMENT 5
- 5.2 WELL INSTALLATION AND DEVELOPMENT 6
  - 5.2.1 Determination of Aquifer Pumping Test Well Location 6
  - 5.2.2 Aquifer Pumping and Observation Well Development 6
- 5.3 AQUIFER PUMPING TEST PROCEDURES 7
  - 5.3.1 Aquifer Pumping Test Design 7
  - 5.3.2 Step-Drawdown Tests 8
  - 5.3.3 Constant-Rate Discharge Tests 9
- 5.4 METHODS OF ANALYSIS 10

6.0 DOCUMENTATION 11

7.0 REFERENCES 12

LIST OF FIGURES

FIGURE 1 AQUIFER PUMPING TEST DATA SHEET 14

3.0

INTRODUCTION

This standard operating procedure (SOP) discusses equipment and procedures that will be used at the Lamefoot Mine Site to conduct aquifer pumping tests in completed wells. This SOP addresses "step-drawdown" and "constant rate discharge" pumping tests.

This SOP serves as a supplement to the site-specific Hydrologic Monitoring Plan (HMP) and is intended to be used together with the other SOPs. The step-by-step procedures described herein are sufficiently detailed to allow field personnel to properly perform aquifer pumping tests.

An aquifer pumping test is an *in situ* test used to assess hydraulic characteristics representative of the aquifer being tested. This is accomplished by hydraulically stressing the aquifer through the removal of water and measuring hydrostatic pressure response. Typically, a production well is pumped at fractions of full capacity with pumping rates changing in a time step fashion, "step-drawdown", and/or at a constant rate, with water levels measured at frequent intervals in the production well and one or more observation wells (Walton 1987). Time-drawdown and distance-drawdown data are recorded and used to calculate aquifer characteristics through type-curve matching, straight-line matching or inflection-point techniques.

4.0

RESPONSIBILITIES AND QUALIFICATIONS

The Project Manager (Echo Bay's or consultant's) is responsible for assuring that aquifer pump testing is conducted according to this procedure. The Project Manager will designate qualified project staff to complete this procedure.

Personnel conducting the aquifer pumping test will be hydrogeologists, geologists and/or engineers with backgrounds in hydrology, or field technicians with an appropriate amount of applicable experience or on-the-job training under the supervision of other qualified personnel.

The designated project staff are responsible for conducting pump testing according to this procedure. They report their progress, and any problems, to the Task Leader or Project Manager. Staff members are responsible for understanding the activities assigned to them and the quality assurance requirements associated with the activities. Problems related to pump testing are also the responsibility of the Task Leader.

5.0
PROCEDURE

5.1 EQUIPMENT

Aquifer pumping tests will be conducted using an arrangement of equipment that will satisfy the requirements specific to the test conditions. The following is a general list of equipment needed to perform aquifer pumping tests:

- A reliable power source
- Submersible or suction pump
- Flow meter
- Discharge control equipment (e.g., gate valve)
- Water discharge line
- Temporary holding tank
- Pressure transducers
- Data loggers
- Electronic water level indicator
- Timeclock or watch with second hand or digital readout of seconds
- Field notebook and/or field forms

Specific equipment and materials requirements will vary with the type of aquifer pumping test being performed. In addition, if the groundwater being extracted is contaminated, contingencies may have to be made for treating the water prior to discharging or disposal.

Guidelines presented in SOP No. 1, Equipment Decontamination, are to be followed for decontaminating equipment involved in aquifer pumping tests.

5.2 WELL INSTALLATION AND DEVELOPMENT

All wells should be installed per guidelines in SOP No. 2, Monitoring Well and Piezometer Installation, except for specific design requirements as set forth in this SOP. A water level measurement will be taken according to SOP No. 5, and well development activities will continue. The water level measurement along with the total well depth measurement will be used to determine the volume of water in the well casing. Well casing calculations are presented in Section 5.4.3 of SOP No. 6, Monitoring Well Sampling.

5.2.1 Determination of Aquifer Pumping Test Well Location

When selecting a site for an aquifer pumping test, consideration will be given to the characteristics of the site as it relates to possible potentiometric surface fluctuations as which may result from nearby pumping or surface loading (traffic, etc.), and to the expected gradient of the water table.

5.2.2 Aquifer Pumping and Observation Well Development

Well development is the process of removing drilling fluids, sediment and smeared or built-up materials from the borehole walls, filter packs, and/or well screens. It also is a process whereby fines within the formation adjacent to the borehole can be removed, thus enhancing well efficiency and allowing the inflow of physically and chemically representative groundwater from the formation adjacent to the screened interval of the well. This process will be used for all observation and pumping wells, whether they are old wells or newly installed wells. Well development procedures are described in SOP No. 2, Monitoring Well and Piezometer Installation.

5.3 AQUIFER PUMPING TEST PROCEDURES

The decision to conduct an aquifer pumping test will be based upon preliminary or available estimates of hydraulic parameters (e.g., hydraulic conductivity, specific capacity, etc.) that are representative of the aquifer. A test will be designed to give the desired information while minimizing the complexity and time required to conduct the test.

Detailed geologic and hydrogeologic information of the test area will be gathered and synthesized into the aquifer pumping test plan. A sufficient amount of time to conduct the test should be considered.

Prior to the beginning of any aquifer pumping test, the following must be assessed:

- Design of Test
- Desired Information
- Geologic Conditions
- Analytical Solution and Boundary Conditions

5.3.1 Aquifer Pumping Test Design

To increase the probability that a proposed test site and associated equipment will yield acceptable results, and to minimize uncertainties in data collection and analysis, the following pumping test design criteria must be evaluated, to the extent that available data allows:

- The extraction well diameter, well depth and the position and length of all well screen intervals open to the aquifer system.

- Radial distance and direction from the extraction well to each observation well and from any interfering boundary (e.g., a stream or lake).

- Radial distance and direction from any known boundaries to each observation well

- Depth to, thickness of, and areal and vertical limits of the aquifer system

- The order of magnitude of pertinent aquifer system hydraulic characteristics (Walton 1987)

5.3.2 Step-Drawdown Tests

A step-drawdown test (Driscoll 1986) is used to assess well performance under conditions of turbulent flow. Also, the data obtained from a step-drawdown test can be used to determine the specific capacity of a well. This information can be used to select the optimum pumping rate and pump-setting depth for a constant-rate discharge test. Preliminary transmissivity and storativity values for the aquifer can also be estimated.

Once the pump has been installed in the production well, and the discharge control equipment, flow meter, pressure transducers, and other previously mentioned equipment have been installed, static water levels at each of the production and observation wells will be collected and recorded. After this has been performed, the data logger will be programmed, pre-test water level data recording initiated, and the test is ready to begin.

During a step-drawdown test, the production well will be pumped at several successively higher pumping rates during which time-drawdown data will be collected for each rate, or step. Generally, a step-drawdown test should be completed within one day, however, in materials of low hydraulic conductivity, effects due to well bore storage may significantly increase the time required to complete the test. The pumping times for each step of the test will be the same (approximately 0.5 to 2 hours in duration) in order to simplify calculations. During the performance of these tests, between three and eight pumping steps will be used.

Electronic pressure transducers and a data logger will be installed in the production well and the observation wells to record time-drawdown data during both the pumping and recovery phases of the test. In addition, manual measurements using an electronic water level indicator will be collected as backup data, and flow rate data also will be collected. The information collected during the duration of this test will be recorded on the data sheet provided in Figure 1.

5.3.3 Constant-Rate Discharge Tests

A constant-rate discharge test, with one or more observation wells, can be used to assess the drawdowns in a well at future times and different discharges, the radius of the cone of influence for individual or multiple wells, as well as the transmissivity and storativity of the aquifer. The pumping rate to be maintained for the duration of the test, the pump-setting depth, and the test duration will be determined prior to commencement of a constant-rate discharge test.

Once the pump has been installed in the production well, the discharge control equipment, flow meter, pressure transducers, and other previously mentioned equipment are installed, static water levels at each of the production and observation wells will be collected and recorded. After this has been performed, the data logger will be programmed, pre-test water level data recording initiated, and the test is ready to begin.

The pump being utilized for the pumping test will begin pumping at a previously determined time. The data logger will be set to begin collecting data just before or at the same time the pump begins pumping. Once the pumping test begins, hand collected time-drawdown measurements will begin. These measurements should begin at the production well and continue at the observation wells in a pattern that will allow for the most rapid collection of data from well to well.

During the pumping test, the flow meter will be monitored regularly to ensure that a constant flow rate is being maintained. In the event that a constant flow rate is not being maintained, adjustments to the pump, power source, or discharge control equipment may need to be performed. Flow rates will be monitored at regular time intervals throughout the duration of the test, with the time intervals being small at the beginning of the test. Records of flow rate versus time and time-drawdown data will be maintained throughout the duration of the test.

After completion of the pumping test, water level recovery data will be collected. The procedures for data collection during the recovery phase of the test will be identical to those preformed during the pumping phase of the test. The information collected during the duration of this test will be recorded on the data sheet provided in Figure 1.

5.4 METHODS OF ANALYSIS

For each aquifer pumping test, the analytical solution will be consistent with the conditions at the test site, the information desired, and the test design. The project hydrogeologist will be responsible for choosing the analytical method and interpreting the test results.

Analytical solutions of pumping test results are based upon the type of flow encountered during the test and the type of aquifer being tested. Analysis of the pumping test results may be facilitated by commercially available computer software for aquifer test analysis.

6.0
DOCUMENTATION

Documentation for this SOP will include the Aquifer Pumping Test Data Sheet (Figure 1) and entries in the field logbook as appropriate.

7.0
REFERENCES

Bear, J. 1979. Hydraulics of Groundwater. McGraw-Hill, New York. 567 p.

Driscoll, F.G. 1986. Groundwater and Wells. Johnson Division. St. Paul, Minnesota. 1,089 p.

Walton, W.C. 1987. Groundwater Pumping Tests. National Water Well Association. Lewis Publishers, Inc. Chelsea, Michigan. 201 p.

FIGURE 1

AQUIFER PUMPING TEST DATA SHEET

DATE ___\\___\\___ PERSON RECORDING DATA ________________

WELL # ____________
HYDROSTRATIGRAPHIC UNIT ______________________
SCREENED INTERVAL ______ft to ______ft

STATIC WATER LEVEL ______ft PUMPING WELL I.D. __________in

DISTANCE TO PUMPING WELL _______ft

TEST START TIME _____:____:____

ELAPSED TIME (Units)	WATER LEVEL (Units)	Q (pumping well) (Units)

.

.

.

etc.

APPENDIX J-5
SOP NO. 5 - WATER LEVEL MEASUREMENT

1.0
TITLE PAGE

STANDARD OPERATING PROCEDURE NO. 5

WATER LEVEL MEASUREMENT

LAMEFOOT MINE SITE

2.0
TABLE OF CONTENTS

Section Page

1.0 TITLE PAGE 1

2.0 TABLE OF CONTENTS 2

3.0 INTRODUCTION 3

4.0 RESPONSIBILITIES AND QUALIFICATIONS 4

5.0 WATER LEVEL MEASUREMENT PROCEDURE 5

- 5.1 DISCUSSION 5
- 5.2 EQUIPMENT LIST 5
- 5.3 MEASUREMENT PROCEDURE 6
- 5.4 DECONTAMINATION 7
- 5.5 DOCUMENTATION 7

6.0 SURFACE WATER FLOW MEASUREMENT 8

LIST OF TABLES

TABLE 1 LARGE 60 DEGREE TRAPEZOIDAL FLUME FREE-FLOW DISCHARGE CONVERSION TABLE 9

LIST OF FIGURES

FIGURE 1 FIELD DATA SHEET FOR WATER LEVEL MEASUREMENT 10

3.0

INTRODUCTION

The purpose of this document is to define the standard procedure for measuring water levels in wells at the Lamefoot Mine Site. This Standard Operating Procedure (SOP) serves as a supplement to the site-specific Hydrologic Monitoring Plan (HMP) and is intended to be used with the other SOPs. This procedure describes equipment and field procedures necessary to collect water level measurements. The well locations and frequency of measurement are specified in the HMP. SOP No. 1 describes decontamination procedures which are applicable to this SOP.

4.0

RESPONSIBILITIES AND QUALIFICATIONS

The Project Manager (Echo Bay's or consultant's) is responsible for assuring that water level measurements are conducted according to this procedure. The Project Manager will designate qualified project staff to complete this procedure.

The designated project staff are responsible for conducting water level measurements according to this procedure. They report their progress, and any problems, to the Task Leader or Project Manager. Staff members are responsible for understanding the activities assigned to them and the quality assurance requirements associated with the activities. Problems related to water level measurement activities are also the responsibility of the Task Leader.

5.0
WATER LEVEL MEASUREMENT PROCEDURE

5.1 DISCUSSION

Water level measurements are used to construct potentiometric surface maps. Therefore, all water level measurements at a given site should be collected within a 24-hour period. The device used to measure water levels should be adequate to attain an accuracy of 0.01 feet. Water levels should be allowed to stabilize for a minimum of 48 hours after well construction and development before measurements are taken.

5.2 EQUIPMENT LIST

The equipment necessary to measure water levels includes:

- Electronic water level indicator capable of producing measurements to a precision of 0.01 feet
- Two 5-gallon buckets or equivalent for decontamination
- Sponges for decontamination
- Field data sheets
- Field notebook
- Paper towels
- Alconox detergent
- Potable water
- Deionized organic-free water
- Appropriate health and safety equipment

5.3 MEASUREMENT PROCEDURE

This section gives the steps to follow when measuring water levels.

- Before any measurement is taken, the water level probe shall be decontaminated. Decontamination procedures are discussed in SOP No. 1 and Section 5.4 of this SOP.

- Make sure the monitoring well is labeled and the location ID is visible on the protective casing.

- After opening the well cover, measure the depth of the static water level and the total depth of the well using an electronic water level indicator. The measuring point for all the wells shall be the top of PVC or steel well casing. The measuring point will be marked by a notch or other mark in the PVC or steel casing. If no mark is present, measure from the north side of the casing. Record measuring point along with measurements in the field data sheet and in the field logbook.

- The static water level and the depth of the well shall be measured with the probe, written down on the field data sheet or field notebook, and immediately rechecked before the probe is removed from the well.

- All columns of field data sheets shall be completed, including time of measurement. A field data sheet for water level measurement is shown in Figure 1. Section 5.5 describes the documentation required.

- Care should be taken to verify the readings during each water level measurement period. Note any significant changes in water level, by comparing the most recent measurement with past measurements, if appropriate.

5.4 DECONTAMINATION

The water level indicator will be decontaminated before using between wells and at the conclusion of measurements. The probe will be decontaminated according to the procedure for decontamination of sampling equipment described in SOP No. 1, Equipment Decontamination. Probe decontamination can be completed at the wells.

5.5 DOCUMENTATION

This section describes the documentation necessary for water level monitoring.

A copy of the field data sheet is shown in Figure 1. It shall be completed during each measuring event. The field data sheets will include date, time, well number, water level, and static water elevation data. A field notebook will also be kept during water level measurement activities, describing decontamination procedures, monitoring procedures, and other observations during water level measurement. Both the data sheets and notebook should be neat and legible, and shall be signed and dated by the person completing the page.

The water level depth below the measuring point (in feet) will be subtracted from the measuring point elevation to determine the elevation of the static water level. The resulting elevation should be checked in the field to see that it is reasonable and the subtraction was performed correctly. If there is a discrepancy, the well should be measured again.

6.0

SURFACE WATER FLOW MEASUREMENT

Surface water flow from Spring 2605 will be measured using a large 60 degree V-shaped trapezoidal flume. This flume has a capacity of measuring from 0.01 to 0.2 cubic feet per second (cfs) (4.5 to 90 gallons per minute (gpm)) of flow.
Flow measurement is obtained by reading the staff gage which is installed on the trapezoidal flume. The staff gage should be read to the nearest 0.01 foot. The staff gage reading is converted to flow by utilizing the relationship in the attached table. Staff gage readings should be recorded in a field notebook.

The flume should be free of debris during flow measurement. Also, it should be noted whether or not all flow upstream is conveyed by the flume, ie. no flow is bypassing or overtopping the flume. In addition, it should be noted if tailwater conditions (backwater) may be affecting flow in the flume and, therefore, discharge measurements.

TABLE 1

LARGE 60 DEGREE TRAPEZOIDAL FLUME FREE-FLOW DISCHARGE CONVERSION TABLE

FIGURE 1

FIELD DATA SHEET FOR WATER LEVEL MEASUREMENT

Instrument Number: ____________________ Date: ____________________________

Sampler's Name: ______________________ Project No.: ______________________

Project Name: ______________________

Location	Date/ Time	Measurement and Location Point Elevation (MSL)	Depth to Water	Depth of Well	Water Elevation (msl)	Sampler's Initials

Note: BE SURE TO DECONTAMINATE THE PROBE BETWEEN WELLS

APPENDIX J-6
SOP NO. 6 - MONITORING WELL SAMPLING

1.0
TITLE PAGE

STANDARD OPERATING PROCEDURE NO. 6

MONITORING WELL SAMPLING

LAMEFOOT MINE SITE

2.0
TABLE OF CONTENTS

Section Page

1.0 TITLE PAGE 1

2.0 TABLE OF CONTENTS 2

3.0 INTRODUCTION 4

4.0 RESPONSIBILITIES AND QUALIFICATION 5

5.0 PROCEDURES FOR MONITORING WELL SAMPLING 6

5.1 EQUIPMENT LIST 6
5.2 SAMPLING PROCEDURE 8

5.2.1 Equipment Decontamination 8
5.2.2 Instrument Calibration 9
5.2.3 Well Evacuation 9
5.2.4 Obtain Water Samples 11
5.2.5 Field Quality Assurance/Quality Control Procedures and Samples 13

5.3 SAMPLE HANDLING 15
5.4 DOCUMENTATION 15

5.4.1 Field Sampling Data Sheet 15
5.4.2 Field Notes 16
5.4.3 Well Volume Calculations 16

6.0 CALIBRATION 18

6.1 pH METER 18
6.2 CONDUCTIVITY METER 19

TABLE OF CONTENTS (Concluded)

LIST OF FIGURES

FIGURE 1 FIELD SAMPLING DATA SHEET FOR GROUNDWATER SAMPLES 20

3.0

INTRODUCTION

The purpose of this document is to define the standard procedure for collecting groundwater samples from wells at the Lamefoot Mine Site. This Standard Operating Procedure (SOP) serves as a supplement to the site-specific Hydrologic Monitoring Plan (HMP) and is intended to be used with the other SOPs. This procedure gives descriptions of equipment, field procedures, and QA/QC procedures necessary to collect groundwater samples from wells using a bailer. The sample locations are specified in the HMP.

Sample identification, labeling, handling, and chain-of-custody procedures are described in SOP No. 7. SOP No. 7 also includes the listing of sample containers, preservatives, and holding times applicable to samples collected using this SOP. SOP No. 1 describes decontamination procedures which are also applicable to this SOP.

4.0
RESPONSIBILITIES AND QUALIFICATIONS

The Project Manager (Echo Bay's or consultant's) is responsible for assuring that monitoring well sampling is conducted according to this procedure. The Project Manager will designate qualified project staff to complete this procedure.

The designated project staff are responsible for conducting monitoring well sampling according to this procedure. They report their progress, and any problems, to the Task Leader or Project Manager. Staff members are responsible for understanding the activities assigned to them and the quality assurance requirements associated with the activities. Problems related to monitoring well sampling are also the responsibility of the Task Leader.

5.0
PROCEDURES FOR MONITORING WELL SAMPLING

5.1 EQUIPMENT LIST

Sample bottles with preservatives added will be obtained from the analytical laboratory. Several extra sample bottles will be obtained in case of breakage and for QA/QC samples.

Equipment that may be used during well evacuation:

- Well keys
- Submersible pump
- Portable generator (or battery) for submersible pump
- PVC discharge hose
- Electronic water level probe
- Assorted tools (knife, screwdriver, etc.)
- Thermometer
- pH meter (with automatic temperature compensation)
- Specific conductivity meter
- Plastic squeeze bottle filled with deionized water
- Chemical-free paper towels or Kimwipes
- Calculator
- Field notebook
- Field data sheet
- Organic-free deionized water
- Waterproof pen
- Paper towels
- Garden hose (if appropriate)

- 5-gallon bucket (for determining flow rate)
- Disposable transparent PVC bailer
- 55-gallon drum or holding tank for storing purge water
- Appropriate health and safety equipment

Equipment that may be used during well sampling with a bailer:

- Electronic water level measurement probe
- Teflon bailer (double check valve)
- Nylon rope or twine
- Thermometer
- pH meter (with automatic temperature compensation)
- Specific conductivity meter
- Plastic squeeze bottle filled with deionized water
- Sample bottles
- Cooler with ice
- Sample labels
- Concentrated nitric acid (for sample bottles if not preserved)
- Grease pencil
- Strapping tape
- Stainless steel bowl for collecting rinsate sample

Equipment used during sample filtration:

- Disposable Nalgene filterware with 0.45-micron filter
- Tygon tubing
- Nalgene hand pump or peristaltic pump

Equipment used during decontamination:

- Alconox detergent (or equivalent)
- Potable tap water
- Deionized water
- Decontamination buckets
- Plastic brushes
- Nitric acid
- Garden-type sprayer filled with deionized organic-free water
- Paper towels
- Plastic sheeting

5.2 SAMPLING PROCEDURE

This section gives the step-by-step procedures for well evacuation and collecting water samples with a bailer. Observations made during sample collection should be recorded in the field notebook and field data sheet as specified in Section 5.4 of this SOP.

5.2.1 Equipment Decontamination

Before any evacuation or sampling begins, all well probes, bailers, and other sampling devices shall be decontaminated. If dedicated equipment is used, it should be rinsed with deionized water. Mobile decontamination supplies will be provided so that equipment can be decontaminated in the field. Each piece of evacuation and/or sampling equipment shall be decontaminated before sampling operations and between each well. The decontamination solutions shall be replaced with clean solutions between each well. A more in depth discussion of equipment decontamination is contained in SOP No. 1, Equipment Decontamination.

5.2.2 Instrument Calibration

Electronic equipment used during sampling includes a pH meter with temperature scale, a conductivity meter, and a water level measurement probe. Before going into the field, the sampler shall verify that all of these are operating properly. The pH and conductivity meters require calibration prior to use every day. Calibration times and readings will be recorded in a notebook to be kept by the field sampler. Specific instructions for calibrating the instruments are given in Section 6.0 of this SOP.

5.2.3 Well Evacuation

The purpose of well purging is to remove stagnant water from the well and obtain representative water from the geologic formation being sampled while minimizing disturbance to the collected samples. Before a sample is taken, the well will be purged until a minimum of three well casing volumes have been removed and field parameters have stabilized, or until the well is pumped dry. Wells shall be purged on the same day that samples are taken. Evacuated well water shall be contained in a 55-gallon drum or holding tank for later disposal.

Before well purging begins, the following procedures are to be performed at each well:

- Note the condition of the outer well casing, concrete well pad, protective posts (if present), and any other unusual conditions of the area around the well.

- Open the well.

- Note the condition of the inner well cap and casing.

- Measure (to nearest 0.01 foot) and record depth of static water level from the measuring point on the well casing and indicate time. Record what the measuring point is (e.g., notch on north side, top of PVC well casing).

- Measure and record total depth of well from the same measuring point on the casing.

- Calculate volume of water in the well casing in gallons based on feet of water and casing diameter. (see Section 5.4.3 for calculation of volumes.)

- From the above calculation, calculate the three casing volumes to be evacuated.

- Determine flow rate of pump in gallons per minute using the 5-gallon bucket or 55-gallon drum (i.e., 5 gallons in 30 sec. = 10 gallons/min.)

- Divide the total volume (gallons) by the flowrate (gallons/minute) to obtain the number of minutes to evacuate three casing volumes.

- Obtain an initial sample from the discharge hose for field measurements (temperature, conductivity, and pH measurements) and observation of water quality.

- Evacuate three volumes of water from the casing and collect water in 55-gallon drums or holding tank for later disposal. Take temperature, specific conductance, and pH measurements after evacuation of each well volume to confirm that the water chemistry has stabilized. Generally, pH values within ±0.2 pH and conductivity within ±10 percent between consecutive readings indicate good stability of the water chemistry. If the chemistry is not stable, continue purging, measuring pH and specific conductance after each one-half well volume.

- Monitor discharge rate and maintain a steady flow.

- If the well is pumped dry during evacuation, it can be assumed that the purpose of removing three well volumes of water has been accomplished, that is, removing all stagnant water which had prolonged contact with the well casing or air. If recovery is very slow, samples may be obtained as soon as sufficient water (approximately 80 percent of original casing volume) is available.

5.2.4 Obtain Water Samples

Obtain samples for chemical analysis within two hours after purging is completed. For slow recovering wells, the sample shall be collected immediately after a sufficient volume is available. The water quality samples shall be taken from within the well screen interval. The following sampling procedure is to be used at each well:

1. Assemble decontaminated Teflon bailer. New nylon rope will be used for sampling the well. Assemble the filtering apparatus.

2. Fill out sample labels as described in SOP No. 7 and affix to sample bottles.

3. Lower the bailer slowly and gently into contact with the water in the well. Lower the bailer to the same depth in the well each time, within the screened interval.

4. Retrieve the bailer smoothly and empty the water in a slow steady stream into containers, with as little aeration as possible.

- The individual sample bottles should be filled in the order listed below:
    - 2 - 16 oz. poly bottles, 1 each for total recoverable and dissolved metals
    - 4 oz. glass jar for pH
    - 1 liter poly bottle for alkalinity, specific conductance, total suspended solids, and total dissolved solids
    - 1 - 500 ml bottle (containing H_2SO_4 preservative) for ammonia and nitrate + nitrite
    - 1 - 1 ℓ HDPE for chloride, fluoride, and sulfate
    - 1 - 500 ml HDPE for nitrate and orthophosphate
- Fill sample bottles for all parameters to almost full and cap quickly.

5. If dissolved metals are required, slowly pour 1 ℓ of the sample through a .45 micron filter. Filter the sample and transfer it to a sample bottle with preservative and cap.

6. For sample bottles that are less than full, mark volume level on bottle with a grease pencil.

7. Place samples on ice in a cooler.

8. Following sampling, measure and record depth of static water level.

9. Record time of sampling.

10. Replace and lock well cap.

11. Complete field documentation.

Note that preservation techniques are discussed in detail in SOP No. 7.

5.2.5 Field Quality Assurance/Quality Control Procedures and Samples

The well sampling order will be determined prior to sampling. Quality assurance/quality control (QA/QC) samples will be collected during groundwater sampling.

QA/QC samples are designed to help identify potential sources of sample contamination and evaluate potential error introduced by sample collection and handling. All QA/QC samples are labeled with QA/QC identification numbers and sent to the laboratory with the other samples for analyses. Specific QA/QC samples that will be collected for the project are described in the HMP.

Rinsate Samples

An equipment rinsate sample of sampling equipment is intended to check if decontamination procedures have been effective. For well sampling with a bailer, a rinsate sample will be collected from the decontaminated sampling bailer prior to sampling. Organic free deionized water will be rinsed through the decontaminated Teflon bailer and used to fill the sample bottles. The same parameters that are being analyzed in the groundwater samples will be analyzed in the rinsate samples. The rinsate sample is assigned a QA/QC sample identification number, stored in an iced cooler, and shipped to the laboratory.

Duplicate Samples

Duplicate samples are samples collected side-by-side to check for the natural sample variance and the consistency of field techniques and laboratory analysis. For groundwater sampling with a bailer, a duplicate sample will be collected at the same time as the primary sample. The primary sample bottles will be filled first, then the duplicate sample bottles until all necessary sample bottles for both the primary sample and the duplicate sample have been filled. The duplicate groundwater sample will be handled in the same manner as the primary sample. The duplicate sample will be assigned a QA/QC identification number, stored in an iced cooler, and shipped to the laboratory.

Field Blanks

Field blanks check for contamination of samples due to factors at the well site. For a field blank, the same parameters that are being analyzed in the groundwater samples will be analyzed. Sample bottles are taken empty to the field and filled at the well site with organic-free deionized water at the time the well is sampled. The samples will be assigned a QA/QC identification numbered, stored in an iced cooler, and shipped to the laboratory with the other samples.

Matrix Spikes

Matrix spikes are used to determine long-term precision and accuracy of the analytical method on various matrices. For this procedure duplicate samples are collected at a well and spiking is done by the lab. Samples are labeled as matrix spikes/matrix spike duplicate (MS/MSD) for the lab. Collect both the matrix spike and matrix spike duplicate at the same well.

5.3 SAMPLE HANDLING

Sample containers and preservatives are specified in SOP No. 7, Sample Handling, Documentation and Analysis. Samples will be labeled and handled as described in SOP No. 7. The parameters for analysis are also specified in SOP No. 7 and in the HMP.

5.4 DOCUMENTATION

5.4.1 Field Sampling Data Sheet

A field sampling data sheet for groundwater samples (Figure 1) will be completed at each sampling location. The data sheet will be completely filled in. If items on the sheet do not apply to a specific location, the item will be labeled as not applicable (NA). The information on the data sheet includes the following:

Well number
Date and time of sampling
Person performing sampling
Water level, depth to bottom of well
Volume of water evacuated before sampling
Conductivity, temperature, and pH during evacuation (note number of well volumes)
Number of samples obtained for each analyses
Container types
Preservation of samples
Record of any QA/QC samples from well
Any irregularities or problems which may have a bearing on sampling quality

5.4.2 Field Notes

Field notes shall be kept in a bound field notebook. The following information will be recorded using water proof ink:

Names of personnel
Weather conditions
Date and time of sampling
Location and well number
Condition of the well
Decontamination information
Initial static water level and total well depth
Calculations (e.g., calculation of evacuated volume)
Calibration information
Sampler's signature

5.4.3 Well Volume Calculations

The following equations shall be used to calculate the volume of water to be removed during well evacuation:

For 2-inch wells:

1 Well Casing Volume (gal) = Total Depth(ft) - Water Level Depth(ft) x 0.163 gal/ft

For 4-inch wells:

1 Well Casing Volume (gal) = Total Depth(ft) - Water Level Depth(ft) x 0.653 gal/ft

For 6-inch wells:

1 Well Casing Volume (gal) = Total Depth(ft) - Water Level Depth(ft) x 1.469 gal/ft

Multiply the volume of one well casing volume by three (3) to obtain the minimum volume of water to be evacuated.

6.0
CALIBRATION

6.1 pH METER

The pH meter must be calibrated each day prior to well evacuation. Calibration and operation of the pH meter should follow the manufacturer's specific instructions. In general, calibration is done by adjusting the meter with standard buffers that bracket the expected pH of the groundwater. Calibration will consist of the following general procedures:

1. Adjust the reading of the pH meter with the electrode placed in the pH 7 buffer by using the calibration knob. Rinse the electrodes with distilled water between buffer adjustments.

2. If the pH is expected to be 7 or above, adjust the reading of the meter with the electrode placed in the pH 10 buffer with the slope (or temperature) knob. If the pH is expected to be 7 or lower, perform the same procedure using the pH 4 buffer.

3. Repeat steps 1 and 2 until the meter gives acceptable readings (±0.1 pH unit) for all the buffers used for calibration.

Note: Always use the same electrode for measurements that was used in the calibration. Recalibrate the meter if the electrode is replaced. Also, the temperature setting on the pH meter often does not match the sample temperature after calibration. The pH readings will be accurate in these cases, provided that the response to the buffers is correct.

Record the time and temperature in the field notebook whenever the pH meter is calibrated.

6.2 CONDUCTIVITY METER

The conductivity meter must be calibrated each day prior to well evacuation. Record time, temperature, and instrument response in the field notebook. Calibration is done by noting the response of the probe to several standard conductivity solutions which bracket the values expected to be measured in the field. Standards of 100, 1000, and 10,000 umhos/cm should be used. If the instrument has a calibration adjustment, set the response to match the standards. Otherwise, simply record the instrument response to each standard in the field notebook.

Note: The probe must be placed in the center of the solution container to prevent disturbance of the electrical field.

FIGURE 1

Field Sampling Data Sheet For Groundwater Samples

Well No.: ____________________ Date: __________
Sampler's Signature: ____________________ Time: __________

Sample Identification: ____________________

Water Level: ____________________
Depth to Bottom of Well: ____________________
Volume to be Excavated: ____________________

Field Parameters

	Before	1st Vol	2nd Vol	3rd Vol	Sample
pH	____	____	____	____	____
Conductivity	____	____	____	____	____
Temperature	____	____	____	____	____
Turbidity	____	____	____	____	____

Containers	Number	Preservatives
________	____	________
________	____	________
________	____	________
________	____	________
________	____	________
________	____	________
________	____	________
________	____	________

QA/QC Samples Collected: ____________________

Comments: ____________________
____
____

APPENDIX J-7

SOP NO. 7 - SAMPLE HANDLING, DOCUMENTATION AND ANALYSIS

1.0
TITLE PAGE

STANDARD OPERATING PROCEDURE NO. 7

SAMPLE HANDLING, DOCUMENTATION AND ANALYSIS

LAMEFOOT MINE SITE

2.0

TABLE OF CONTENTS

Section — Page

1.0 INTRODUCTION 3

2.0 RESPONSIBILITIES AND QUALIFICATIONS 4

3.0 PROCEDURES FOR SAMPLE HANDLING, DOCUMENTATION AND ANALYSIS 5

3.1 SAMPLE LABELING 5
3.2 SAMPLE HANDLING 8
3.3 SAMPLE ANALYSES 8
3.4 SAMPLE DOCUMENTATION 9

LIST OF TABLES

TABLE 1 SAMPLE ANALYTICAL METHODS (METALS)
TABLE 2 SAMPLE ANALYTICAL METHODS (ANIONS AND OTHERS)

LIST OF FIGURES

FIGURE 1 EXAMPLE CHAIN OF CUSTODY FORM

1.0

INTRODUCTION

The purpose of this document is to define the standard procedures for sample handling, documentation, and analysis at the Lamefoot Mine Site. This Standard Operating Procedure (SOP) serves as a supplement to the site-specific Hydrologic Monitoring Plan (HMP) and is intended to be used together with the other SOPs.

2.0

RESPONSIBILITIES AND QUALIFICATIONS

The Project Manager is responsible for assuring that sample handling, decontamination and analysis is conducted according to this procedure. The Project Manager will designate qualified project staff to complete this procedure.

The designated project staff are responsible for conducting sample handling and documentation according to this procedure. They report their progress, and any problems, to the Task Leader or Project Manager. Staff members are responsible for understanding the activities assigned to them and the quality assurance requirements associated with the activities. Problems related to sample handling, documentation or analysis are also the responsibility of the Task Leader.

3.0
PROCEDURES FOR SAMPLE HANDLING, DOCUMENTATION, AND ANALYSIS

3.1 SAMPLE LABELING

All sample labels should be filled out with waterproof ink and numbered. Soil and water sample labels will be supplied by the laboratory. Soil samples collected in stainless steel liners will be capped immediately following collection and a completed label attached. For soil samples collected in jars and sample bottles for groundwater analyses, sample labels should be completed and attached prior to sample collection. Labels may be partially completed prior to sample collection. The date, time, sampler's initials, and the sample identification number should not be completed until the time of sample collection. At a minimum, each label shall contain the following information:

- Project Name
- Grab or composite sample
- Sampler's company affiliation
- Date and time of sample collection
- Analyses required
- Preservation used
- Sampler's initials
- Filtered (if applicable)
- Sample identification (see below)

The generalized sample designation system for groundwater, surface water, and mine water samples (including QA/QC samples) is a combination of a two to four letter and a one- to four-digit number sample location identifier (XXXX-####) and a two-digit number sample identifier (##). For a groundwater sample, the two to four letters of the sample location identifier will describe one of the following types of sample locations:

LF	Existing monitoring well at the Lamefoot Mine Site
NT	New monitoring well at the Lamefoot Mine Site
MSW	Mine supply well at the Lamefoot Mine Site
SP	State park well
HECK	Heckley private well
CORL	Corl private well
PERK	Perkl private well

For a surface water sample, the two to four letters of the sample location identifier will describe one of the following types of sample locations:

SPR	Spring
WET	Wetland

For a mine water sample, the two to four letters of the sample location identifier will describe one of the following types of sample locations:

UGS	Underground seep
PORT	Portal
IP	Infiltration pond

In the case of a groundwater sample, a one- or two-digit number will be used to identify the specific existing or new monitoring well the sample was collected from (e.g., LF1 or NT4). The mine supply well and the remaining private wells will not require an identification number. Surface water samples collected from a spring or wetland area will be identified with a four-digit number, which corresponds to the elevation of the spring or wetland (e.g., SPR2605 is the spring located at elevation 2,605 feet). Mine water samples collected from underground seeps will be numbered sequentially, and their locations will be identified on a map at the time of collection for later reference (e.g., UGS1, UGS2,...). The portal and infiltration pond samples will not require a number identifier.

The two-digit number sample identifier differentiates the type of sample (i.e., groundwater vs. surface water vs. mine water) and whether the sample is the primary sample of a QA/QC sample (i.e., duplicate, rinsate samples, field blank, etc.). The first digit of the sample identifier will be designated depending on the type of sample as follows:

•	1	Groundwater sample
•	2	Surface water sample
•	3	Mine water sample

The second digit of the sample identifier will be designated as follows:

•	0	Primary sample
•	1	Duplicate sample
•	3	Rinsate sample
•	4	Field blank

QA/QC groundwater samples collected for matrix spike/matrix spike duplicate analysis will be labeled as such on the sample containers.

The following are examples of possible sample designation numbers:

NT2-10	(Primary groundwater sample from well NT2)
WET2495-21	(Duplicate surface water sample from wetland at elevation 2,495')

IP-32	(Rinsate sample collected prior to sampling at the infiltration pond)

The generalized sample identification scheme presented above will be modified as necessary to adapt to specific site sampling requirements.

3.2 SAMPLE HANDLING

This section discusses proper sample containers, preservatives, and handling and shipping procedures. Tables 1 and 2 summarize the information contained in this section and also include the sample holding times for each analysis for groundwater samples.

3.2.1 Sample Containers

Certified, commercially clean sample containers shall be obtained from the contract analytical lab. If appropriate, the bottles shall be labeled by the lab to indicate the type of sample to be collected. Required preservatives shall be prepared and placed in the bottles for aqueous analyses at the laboratory prior to shipment to the site.

3.2.2 Sample Preservation

All samples will be stored on ice in an insulated cooler immediately following sample collection. As noted above, sample containers for aqueous samples will be sent by the laboratory containing the appropriate preservatives.

Tables 1 and 2 lists appropriate sample containers for the specific analyses required for this project.

3.2.3 Sample Handling and Shipping

Sample containers will be placed in resealable plastic storage bags and wrapped in protective packing material (if appropriate). Samples will then be placed in a cooler with ice (double bagged using plastic trash bags) for shipment to the laboratory. The drain on the cooler shall be taped shut. Samples collected in glass containers will be packed in foam liners and bubble packing or styrofoam peanuts to ensure that no breakage occurs during shipment.

A completed chain-of-custody form for each cooler will be placed in a ziplock bag and taped to the inside of the cooler lid. Coolers will be wrapped with strapping tape at two locations to secure lids. Dated and signed custody seals shall be placed on the outside of each cooler.

3.3 SAMPLE ANALYSES

The holding time is specified as the maximum allowable time between sample collection and analysis and/or extraction, based on the analyte of interest, stability factors, and preservation methods. Allowable holding times are listed in Tables 1 and 2. Samples should be sent to the laboratory daily after collection.

Chemical constituents which will be analyzed during the field investigation will be identified in the HMP.

3.4 SAMPLE DOCUMENTATION

This section describes documentation required in the field notes and sample Chain-of-Custody requirements.

3.4.1 Field Notes

Documentation of observations and data acquired in the field will provide information on the acquisition of samples and also provide a permanent record of field activities. The observations and data will be recorded with waterproof ink in a permanently bound weatherproof field notebook with consecutively numbered pages and, if applicable, on field sampling data sheets.

The information in the field notebook or on the field sampling data sheets will include the following as a minimum. Additional information is included in the specific SOPs regarding the appropriate data sheets.

- Project name
- Location of sample
- Sampler's signature
- Date and time of sample collection
- Sample identification numbers and sample depth (if applicable)
- Description of samples (matrix sampled), composite or grab sample
- Analysis to be performed
- Number and volume of samples
- Description of QA/QC samples (if collected)
- Sample methods or reference to the appropriate SOP
- Sample handling, including filtration and preservation, as appropriate for samples
- Field observations
- Results of any field measurements, such as depth to water, pH, temperature, and conductivity
- Decontamination information
- Calibration information
- Personnel present

Changes or deletions in the field notebook should be lined out with a single strike mark, initialed and dated by person making change, and remain legible. Sufficient information should be recorded to allow the sampling event to be reconstructed without relying on the collector's memory.

Each page of the field notebook will be signed by the person making the entry. Anyone making entries in another person's field book will sign and date those entries.

3.4.2 Sample Chain-Of-Custody

During field sampling activities, traceability of the sample must be maintained from the time the samples are collected until laboratory data are issued. Information on the custody, transfer, handling, and shipping of samples will be recorded on a Chain-of-Custody (COC) form. An example COC form is shown on Figure 1. The COC is a three-part carbonless form.

The sample handler will be responsible for initiating and filling out the COC form. The COC will be signed by the sampler when the sampler relinquishes the samples to anyone else. A COC form will be completed for each set of samples collected in one cooler, and will contain the following information:

- Sampler's signature and affiliation
- Project number
- Date and time of collection
- Sample identification number
- Sample type/matrix
- Grab or composite sample
- Preservative used
- Analyses requested
- Number of containers

- Signature of persons relinquishing custody, dates, and times
- Signature of persons accepting custody, dates, and times (laboratory)
- Method of shipment (i.e., hand delivered)

The person responsible for delivery of the samples to the laboratory will sign the COC form, retain the last copy of the three-part COC form, document the method shipment, and send the original and the second copy of the COC form with the sample (taped in a ziplock bag to inner cooler lid). Upon receipt at the laboratory, the person receiving the samples will sign the COC form and return the second copy to the Project Manager. Copies of the COC forms and all custody documentation will be received and kept in the central files. The original COC forms will remain with the samples until final disposition of the samples by the laboratory. The analytical laboratory will dispose of the samples in an appropriate manner 60 to 90 days after data reporting. After sample disposal, a copy of the original COC will be sent to the Project Manager by the analytical laboratory to be incorporated into the central files.

TABLE 1

SAMPLE ANALYTICAL METHODS[1]
(Metals)

Parameter	Recommended Analytical Method	Recommended Number of Containers per Sample[2]	Preservation	Holding Time from Date Collected
Aluminum (Al)	EPA Method 200.7	1-L HDPE[3]	4°C and HNO_3 to $pH<2$	6 months
Antimony (Sb)	EPA Method 200.7			6 months
Arsenic (As)	EPA Method 206.2			6 months
Barium (Ba)	EPA Method 200.7			6 months
Beryllium (Be)	EPA Method 200.7			6 months
Cadmium (Cd)	EPA Method 200.7			6 months
Calcium (Ca)	EPA Method 200.7			6 months
Chromium (Cr), Total	EPA Method 200.7			6 months
Copper (Cu)	EPA Method 200.7			6 months
Iron (Fe)	EPA Method 200.7			6 months
Lead (Pb)	EPA Method 239.2			6 months
Magnesium (Mg)	EPA Method 200.7			6 months
Manganese (Mn)	EPA Method 200.7			6 months
Mercury (Hg)	EPA Method 245.1 or 245.2			28 days
Molybdenum (Mo)	EPA Method 200.7			6 months
Nickel (Ni)	EPA Method 200.7			6 months
Potassium (K)	EPA Method 200.7			6 months
Selenium (Se)	EPA Method 270.2			6 months
Silicon	EPA Method 200.7			6 months
Silver (Ag)	EPA Method 200.7			6 months
Sodium (Na)	EPA Method 200.7			6 months

TABLE 1
(Concluded)

Parameter	Recommended Analytical Method	Recommended Number of Containers per Sample[2]	Preservation	Holding Time from Date Collected
Thallium (Tl)	EPA Method 279.2			6 months
Zinc (Zn)	EPA Method 200.7			6 months

[1] EPA Methods are NPDES-approved, per 40 CFR 136 Tables: EPA 200 Methods from Methods for Chemical Analysis of Water and Wastes, EPA 600/4-79-020, March, 1979, Revised March 1983.

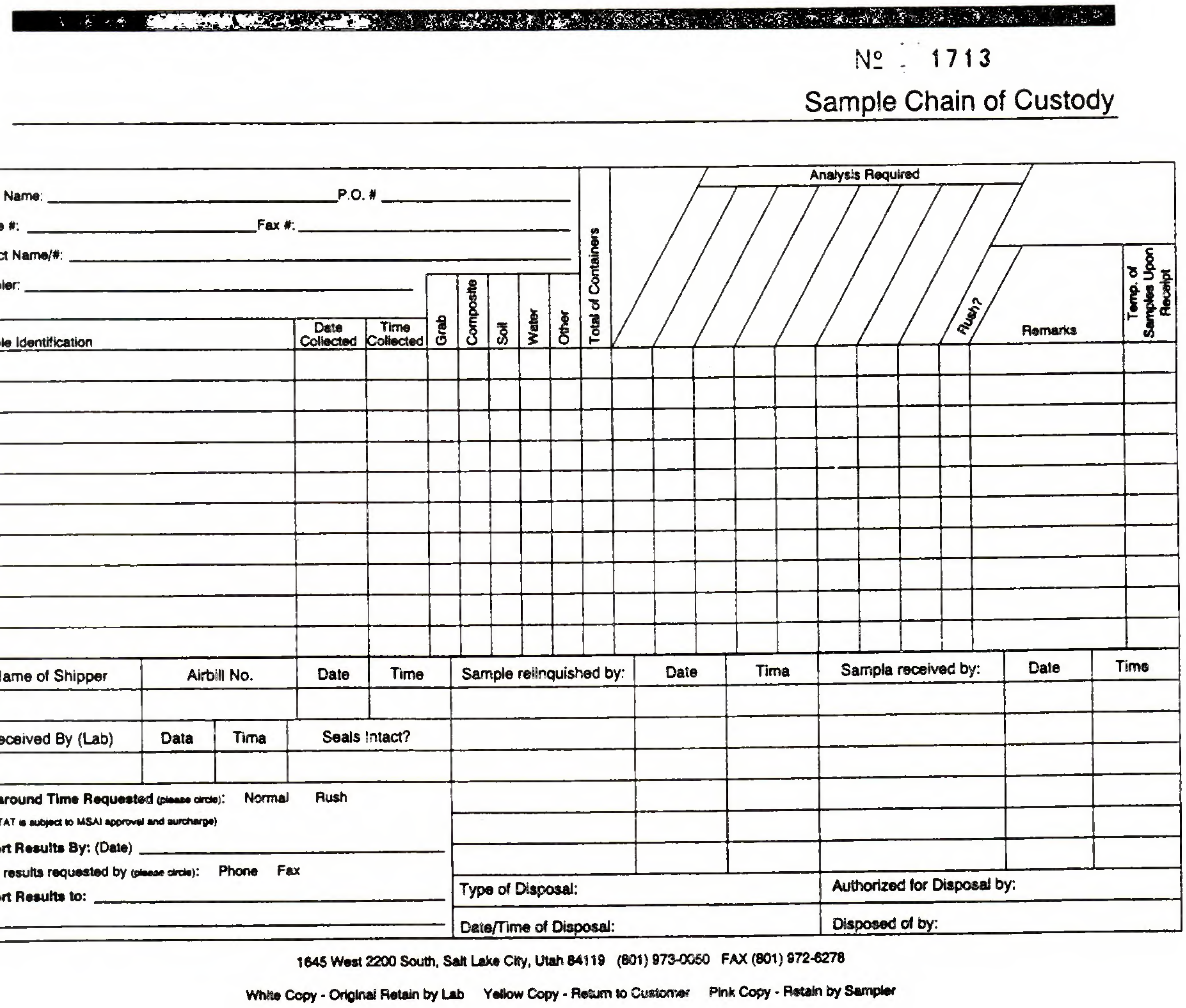
[2] Laboratory to furnish bottles with analysis type labelled and preservative added.

[3] HDPE = High-density polyethylene

NOTE: Samples for metals analyses will be analyzed for total recoverable metals. Selected samples may be field-filtered and analyzed for dissolved metals.

FIGURE 1

Example Chain-of-Custody Form



№ . 1713

Sample Chain of Custody

Client Name: ______________________ P.O. # ____________

Phone #: ____________________ Fax #: ____________________

Project Name/#: ______________________________

Sampler: ______________________________

									Analysis Required										
Sample Identification	Date Collected	Time Collected	Grab	Composite	Soil	Water	Other	Total of Containers									Rush?	Remarks	Temp. of Samples Upon Receipt

Name of Shipper	Airbill No.	Date	Time

Received By (Lab)	Data	Tima	Seals Intact?

Turnaround Time Requested (please circle): Normal Rush

(Rush TAT is subject to MSAI approval and surcharge)

Report Results By: (Date) ______________

Rush results requested by (please circle): Phone Fax

Report Results to: ______________

____

Sample relinquished by:	Date	Tima	Sampla received by:	Date	Time

Type of Disposal:	Authorized for Disposal by:
Date/Time of Disposal:	Disposed of by:

1645 West 2200 South, Salt Lake City, Utah 84119 (801) 973-0050 FAX (801) 972-6278

White Copy - Original Retain by Lab Yellow Copy - Return to Customer Pink Copy - Retain by Sampler